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March 1976

ENERGY CONSERVATION TECHNIQUES
FOR THE IRON FOUNDRY CUPOLA

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ABSTRACT

This report presents the results of an investigation into various existing or emerging technologies which can be utilized to reduce the energy consumption and pollution control costs of typical cupola operation. There were two primary motivating factors behind this investigation. First, the rapid rise in fuel costs has made cupola operation difficult for some foundries. Second, the necessity of installing highly efficient pollution control devices on the cupola has again placed a financial burden upon the smaller foundries.

This report details those options available to the foundries in terms of technological devices which will conserve energy and capital. Included in this investigation were hot blast recuperation, divided blast, oxygen enrichment and innovative pollution control equipment.

This report was submitted by the Flynn and Enrich Company in partial fulfillment of Contract Number 68-02-0286 under the sponsorship of the Environmental Protection Agency.

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Mr. Robert McCrillis of the Officer of Research and Development, U.S. Environmental Protection Agency, Research Triangle Park, North Carolina 27711, served as the Project Officer.

SECTION I

CONCLUSIONS

There exist various types of devices and technologies which can successfully be used to reduce the cost of cupola operation. Hot blast recuperative heat exchangers are now manufactured by several companies. None of these systems appear to experience the problems such as fouling which have detracted from such devices in the past. Hot blast operation can reduce coke consumption by up to 25 percent while reducing the exhaust gas volume to the pollution control device. Divided blast operation can produce results similar to hot blast recuperators with a much lower capital cost. Oxygen enrichment of the combustion air can also affect coke consumption but appears more useful as a combustion control device.

New types of air pollution control devices have also been studied. The three devices included in this report appear to control pollution more economically than the conventional devices when applied to the cupola.

There are many uses open to foundry engineers for waste heat from the cupola, from in-plant heating to sand drying. Applications of waste heat must be evaluated on a plant by plant basis since the plant layout and economic policy are usually the deciding factors in such cases.

SECTION II

RECOMMENDATIONS

Since this report deals with methods of reducing iron foundry operational and air pollution control costs, it is recommended that it be made available to all Environmental Protection Agency personnel who deal with setting compliance schedules for foundries.

It is recommended that a yearly update of this report be made to take into account emerging technologies and that these updates be made available to the foundry industry and compliance officers.

It is also recommended that the EPA and ERDA aid in the design of new foundries to demonstrate new pollution control and energy conservation technologies on a trial basis.

SECTION III

INTRODUCTION

The cupola is the device most often used for the melting of gray iron. It is also generally considered to be the most economical melting technique despite the fact it is an extremely wasteful device from an energy standpoint. Two events in recent years, however, have caused significant difficulties in the use of the cupola. The first event was the Clean Air Act which produced numerous air pollution regulations from federal, state and local agencies. The second event was the substantial increase in fuel prices.

These two occurrences have made the cupola a much more expensive device to operate and maintain than would have been thought possible ten years ago. All three problem areas; energy waste, air pollution control and coke price increase, are part of the difficulty involved in running a successful and economical cupola operation. The adverse impact of each, however, can be minimized by the application of existing technology. It is the purpose of this manual to introduce these various technologies to foundry management. While these technologies are for the most part not new, they have not been utilized extensively by the foundries due to the fact that they were not necessary prior to the environmental movement and the escalation in energy costs.

First the report details the operation of the cupola from an energy viewpoint. Following this is a section devoted to air pollution aspects of the cupola. A review is then made of the fuel situation as it now exists in this country. After defining these problem areas in depth attention is then focused on solutions. Recuperative hot blast, divided blast, oxygen enrichment and innovative pollution control devices are dealt with both from a theoretical viewpoint and from the result of an industry survey. The final section is devoted to outlining options available to the cupola engineer on the basis of the various technologies reviewed.

SECTION IV

CUPOLA OPERATION

The cupola is simply a device for melting iron to a specific chemical composition and at a specific temperature. The physical characteristics of conventional type cupolas are shown in Figure 1. In preparation for melting, the bottom doors of the cupola are propped shut and a layer of sand up to 10 inches in depth placed above them up to the iron tapholes. A layer of coke is placed on top of the sand and ignited usually with natural gas. More coke added to a specified height above the cupola's tuyeres and then alternate charges of coke (mixed with a fluxing agent) and iron are fed into the cupola through the charging door.

Combustion air is forced into the cupola through the tuyeres. The heat generated by combustion melts the iron charge and the molten metal flows through the coke layers to the sand bottom where it can be tapped. For an in-depth explanation of cupola operating practices the reader is referred to the manual entitled "The Cupola and Its Operation" (1)* prepared by the American Foundrymen's Society.

The cupola is an energy intensive device with over 50% of the heat generated being exhausted to the atmosphere. For any good foundry operation an energy balance should be constructed. The procedure for doing such, while somewhat involved, lays the basis for the correct understanding of the cupola. An example of how to perform a heat balance on the cupola is as follows.

ENERGY BALANCE

A. Heat Input

Table 1 and 2 list data from a typical cupola operation. It must be noted that these data are from a relatively stable period of cupola operation and that startup and shutdown data are not included. Other data used in this analysis can be found in Appendix A.

1. Total heat input of coke

$$\begin{aligned}\frac{\text{lbs of carbon burned}}{\text{hr}} &= \frac{\text{lb of coke}}{\text{hr}} \times \frac{\% \text{ fixed carbon}^{**}}{100} \\ &= 6588.2 \times 0.92 \\ &= 6061.1 \text{ lb/hr of carbon}\end{aligned}$$

*Numbers in brackets refer to references which are listed in SECTION X, REFERENCES.

**Refer to Appendix B for English to Metric conversion factors.

MELTING OF GRAY IRON

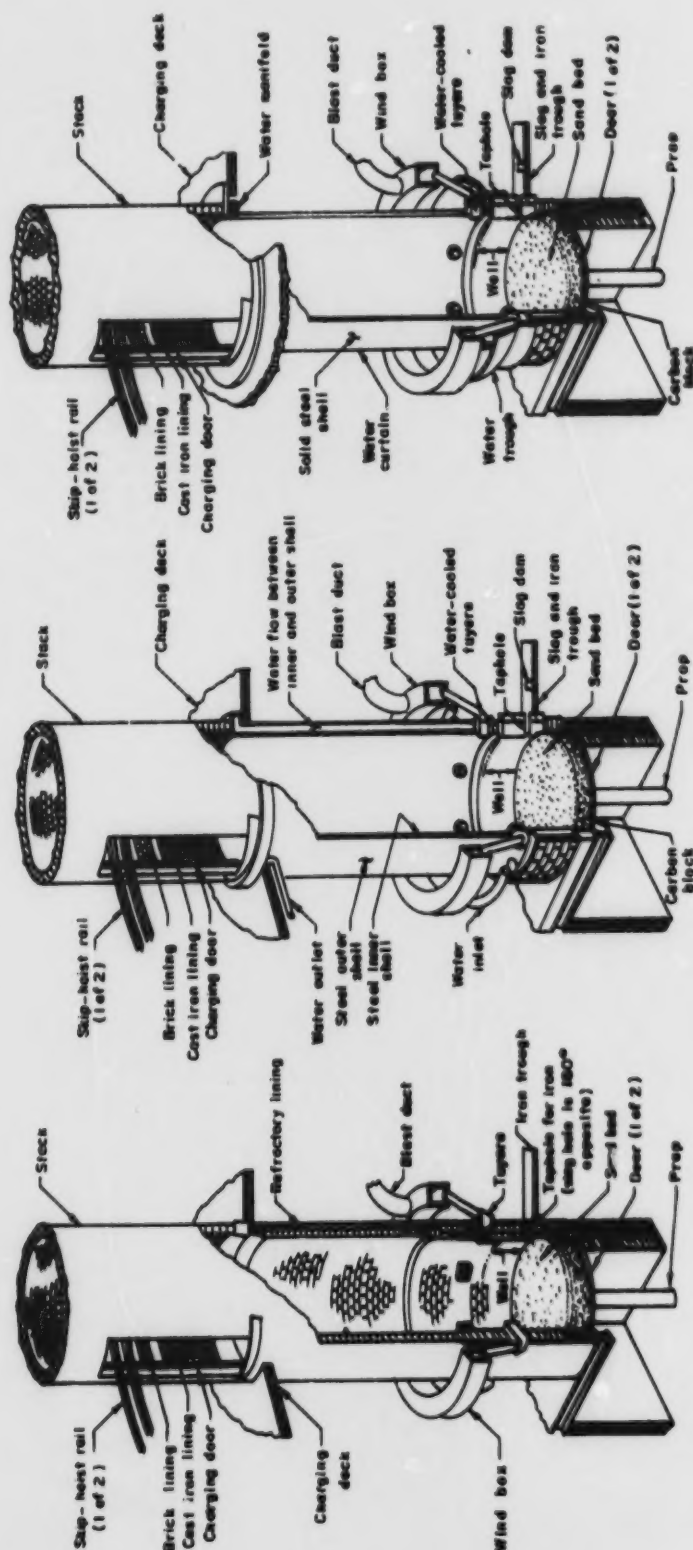


Figure 1. Cupola characteristics.

Since carbon yields 14,452 BTU per lb combusted then the total potential heat is:

$$6061.1 \frac{\text{lb}}{\text{hr}} \times 14,452 \frac{\text{BTU}}{\text{lb}} \text{ of carbon}$$

or 87,595,017 BTU/hr

a. Carbon loss to dissociation

Not all of the carbon is combusted since some of it is lost in the dissociation of water vapor in the combustion air to carbon monoxide and hydrogen. The reaction is as follows:



Using a psychometric chart and the data from Table 1 (i.e., dry bulb temperature, wet bulb temperature) it can be determined that there are approximately 6.18 grains of water per cubic foot of blast air. If it is assumed that the blast air volume given in Table 1 is approximately correct then the total rate of moisture into the cupola is:

$$6.18 \frac{\text{grains}}{\text{ft}^3} \times 678,000 \frac{\text{ft}^3}{\text{hr}} \times \frac{1 \text{ lb}}{7000 \text{ grains}}$$

or 598 lbs of H₂O per hour

From equation 1 it can be seen that 12 lbs of carbon are required to dissociate 18 lbs of water vapor, then

$$\frac{598 \text{ lbs of H}_2\text{O}}{\text{hr}} \times \frac{12 \text{ lbs of carbon}}{18 \text{ lbs of H}_2\text{O}}$$

or 398 lbs/hr of carbon are used in this reaction.

b. Carbon loss to iron

Carbon content of iron charged %:	2.52
Carbon content of iron tapped %:	3.00
Carbon loss %:	0.48

Since the iron charged per hour is 56,000 lbs/hr

$$56,000 \frac{\text{lbs}}{\text{hr}} \times .0048$$

or 268 lbs/hr carbon loss

Table 1. CUPOLA DATA

Charging rate lb/hr	56,000
Coke rate lb/hr	6,588.2
Limestone rate lb/hr	1,120
Blast volume ft ³ /hr	678,000
Blast temperature °F	70
Tapping temperature of metal °F	2,762
CO ₂ Content of stack gas %	12.00
CO Content of stack gas %	14.9
Dry bulb temperature of blast air °F	70
Wet bulb temperature of blast air °F	68
Barometric pressure in.HgA	29.92
Stack temperature °F	900

Table 2. TYPICAL CHEMICAL COMPOSITIONS

Tapped Iron

-Carbon Content %	3.0
-Silicon Content %	0.60
-Manganese Content %	0.48

Charged Iron

-Carbon Content %	2.52
-Silicon Content %	0.70
-Manganese Content %	0.72

Slag

CaO Content %	22.00
FeO Content %	2.5
CaCO ₃ Content of limestone %	98
Fixed Carbon Content of Coke %	92

c. Net heat input from coke:

Carbon input	6,061.1 lb/hr
Dissociation loss	398 lb/hr
Iron loss	268 lb/hr
Net	<u>5,395.1 lb/hr</u>

Net heat input

$$5,295.1 \frac{\text{lbs carbon}}{\text{hr}} \times 14,452 \frac{\text{BTU}}{\text{lbs carbon}}$$

or 77,969,985 BTU/hr

2. Heat from oxidation reactions

a. Pounds per hour of slag

Flux added	1120 lbs/hr
CaCO ₃ content of flux	98%
CaCO ₃ weight	1097.6 lb/hr
CaCO ₃ moles	10.976

Since the same number of moles of CaO are formed as CaCO₃ charged then

CaO in slag	10.976 moles
CaO in slag (at 56 lbs/mole)	614.7 lbs

From Table 3 the CaO content of the slag is 22%.

Therefore, the total amount of slag produced per hour is

$$614.7 \frac{\text{lbs CaO}}{\text{hr}} \times \frac{\text{lbs slag}}{.22 \text{ lbs carbon}}$$

or 2794 pounds slag per hour.

b. Oxidation of iron

FeO content of slag	2.5%
Heat of formation	1,596 BTU/lb
Heat from oxidation:	$2794 \frac{\text{lb}}{\text{hr}} \text{ slag} \times 0.025 \frac{\text{lb FeO}}{\text{lb slag}} \times 1,596 \frac{\text{BTU}}{\text{lb FeO}}$

$$= 111,480 \text{ BTU/hr}$$

c. Oxidation of silicon

% in charge	0.70
% in melt	0.60
% oxidized	0.10
silicon oxidized	56 lbs/hr
heat from oxidation =	$56 \frac{\text{lbs}}{\text{hr}} \times 13,140 \frac{\text{BTU}}{\text{lb}}$
	$= 735,840 \frac{\text{BTU}}{\text{hr}}$

d. Oxidation of manganese

3. Energy from combustion air

In this example the blast air was not preheated therefore, the sensible heat input from the air is not significant.

4. Total heat input in BTU/hr

Potential heat from coke	87,595,017
Carbon loss from dissociation	5,751,896
Carbon loss from iron pickup	3,873,136
Oxidation of iron	111,480
Oxidation of manganese	405,216
Oxidation of silicon	735,840
Total heat input BTU/hr	<u>79,222,521</u>

B. Heat Output

1. Iron melting

Mean specific heat of molten iron	$0.2080 \frac{\text{BTU}}{\text{lb}^\circ\text{F}}$
Molten iron of flowrate	$55,946 \frac{\text{lb}}{\text{hr}}$
Heat content of iron when tapped	$55,946 \frac{\text{lb}}{\text{hr}} \times 0.2080 \frac{\text{BTU}}{\text{lb}^\circ\text{F}} \times 2692^\circ\text{F}$
	or 31,326,180 BTU/hr

2. Slag

a. Sensible heat of slag

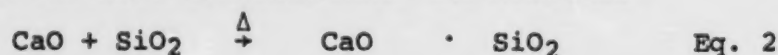
Amount of slag	2,794 lbs/hr
Mean specific heat	$0.320 \frac{\text{BTU}}{\text{lb}^\circ\text{F}}$

$$\text{Heat content} = 2794 \frac{\text{lb}}{\text{hr}} \times 0.3202 \frac{\text{BTU}}{\text{lb}^\circ\text{F}} \times 2692^\circ\text{F}$$

or 2,409,119 BTU/hr

b. Heat of reactions

From the heat of formations the reaction



requires 37,800 BTU. Since 100 pounds of CaCO_3 is required to form one mole of $\text{CaO} \cdot \text{SiO}_2$, the energy required per pound of CaCO_3 is 378 BTU. Since there are 1,097.6 pounds of CaCO_3 (from 2a)

$$1,097.6 \frac{\text{lb}}{\text{hr}} \text{CaCO}_3 \times 378 \text{ BTU/lb}$$

or 414,892.8 BTU/hr

c. Total heat output from slag

Sensible heat of slag	2,409,119 BTU/hr
Heat of reaction	414,892.8 BTU/hr
Total heat content	2,824,001.8 BTU/hr

3. Decompositon of water

Besides tying up some of the carbon in the decomposition reaction Eq. 1 some of the energy liberated by combustion is used in the reaction. Since approximately 2,898 BTU/lb is required

$$2,898 \text{ BTU required} \times 598 \frac{\text{lbs of H}_2\text{O}}{\text{lb H}_2\text{O hr}}$$

or 1,733,004 BTU/hr

4. Stack gas

a. Sensible heat

Knowing the flue gas composition the pounds per hour of each constituent can be calculated. From Table 1

CO_2	12.00% - 0.10 moles
CO	14.9% - 0.182 moles
therefore, N_2	73.2% - 0.718 moles

Since 5,395 lbs of carbon are charged (having subtracted out the appropriate losses) and since

$$\text{CO}_2 + \text{CO} = 0.12 + 0.149 = 0.269 \text{ moles of carbon}$$

or 3.228 pounds of carbon to produce the stated CO and CO₂ stack gas contents then the total moles of stack gas produced are

$$\frac{\text{moles of stack gas} \times 5,395 \text{ lb of carbon}}{3.228 \text{ lb of carbon}} \text{ hr}$$

or 1671.3 moles of stack gas.

The stack gas therefore contains

$$\begin{aligned} 1,671.3 \times 0.12 &= 200.556 \text{ moles of CO}_2 \\ \text{or} &8,824.5 \text{ lbs of CO}_2 \\ 1,671.3 \times 0.149 &= 249.02 \text{ moles of CO} \\ \text{or} &6,972.7 \text{ lbs of CO} \\ 1,671.3 \times 0.732 &= 1223.4 \text{ moles of N}_2 \\ \text{or} &34,255 \text{ lbs of N}_2 \end{aligned}$$

Adding in the H₂ liberated by the dissociation reaction

$$\text{H}_2 = \frac{598 \text{ lbs of H}_2\text{O}}{\text{hr}} \times \frac{2 \text{ lbs of H}_2}{18 \text{ lbs of H}_2\text{O}} = 66.44 \frac{\text{lb}}{\text{hr}}$$

Using the mean specific heats taken from the tables in Appendix B for the various gaseous constituents at 900°F

Heat content of CO₂ =

$$\begin{aligned} 0.2428 \frac{\text{BTU}}{\text{lb}^\circ\text{F}} \times 830^\circ\text{F} \times 8,824.5 \frac{\text{lbs of CO}_2}{\text{hr}} \\ \text{or } 1,778,348.5 \frac{\text{BTU}}{\text{hr}} \end{aligned}$$

Heat content of CO =

$$\begin{aligned} 0.2564 \frac{\text{BTU}}{\text{lb}^\circ\text{F}} \times 830^\circ\text{F} \times 6,972.7 \frac{\text{lbs of CO}}{\text{hr}} \\ \text{or } 1,483,874.2 \frac{\text{BTU}}{\text{hr}} \end{aligned}$$

Heat content of N₂ =

$$0.2544 \frac{\text{BTU}}{\text{lb}^\circ\text{F}} \times 830^\circ\text{F} \times 34,255 \frac{\text{lbs of N}_2}{\text{hr}}$$

or $7,233,011.8 \frac{\text{BTU}}{\text{hr}}$

Heat content of H_2 =

$3.466 \frac{\text{BTU}}{\text{lb}^\circ\text{F}} \times 830^\circ\text{F} \times 66.44 \frac{\text{lb of H}_2}{\text{hr}}$

or $191,133.2 \text{ BTU/hr}$

Total CO_2	1,778,348.5 BTU/hr
CO	1,483,874.2 BTU/hr
N_2	7,233,011.8 BTU/hr
H_2	<u>191,133.2 BTU/hr</u>
Total sensible heat in stack gas	10,686,367.7 BTU/hr

b. Latent heat

Since the heat of combustion of CO is 4,346 BTU/lb then

$6,972.7 \frac{\text{lb of CO}}{\text{hr}} \times 4,346 \frac{\text{BTU liberated}}{\text{lb of CO}}$

or $30,303,354.2 \frac{\text{BTU}}{\text{hr}}$

Total stack gas heat content = sensible + latent =
 $10,686,368 + 30,303,354 = 40,989,722 \text{ BTU/hr}$

5. Radiation from cupola

The radiation from the cupola cannot be adequately measured. It is taken to be the difference between the heat input and output from the cupola.

a. Heat input

79,222,515 BTU/hr

b. Heat output

1. Iron	31,326,180 BTU/hr
2. Slag	2,824,012 BTU/hr
3. Decomposition	1,733,044 BTU/hr
4. Stack gas sensible	10,686,368 BTU/hr
5. Stack gas latent	<u>30,303,354 BTU/hr</u>
Total heat output	76,872,954 BTU/hr

c. Radiation from cupola

2,349,557 BTU/hr

As can be seen in this example 40,989,722 BTU per hour are lost by exhausting the stack gas to the atmosphere. This amounts to approximately 52% of the heat supplied to the system.

There are other energy inputs that should be included in an overall energy balance of a cupola system. For example, there is the energy expended in the charging system and other auxiliary equipment. There are two additional energy inputs which are inherent inputs to the system although they are not usually considered in a normal energy balance. They are the ID fan and blast blower energy requirements and the energy expended by the afterburners. A cupola's fan horsepower requirements could go into the thousands depending on the exhaust volume and pressure drop. As will be explained in the section on air pollution the exhaust flow from a cupola could run five fold over what the blast rate is due to deliberate infiltration. If a venturi scrubber is used for pollution control the system pressure drop would easily exceed fifty inches of water. If we assume a 1500 hp fan this would add another 3,819,600 BTU per hour for a total energy input of 83,042,115 BTU/hr.

Afterburners in a cupola system of the size used in the example usually run 15 to 25 million BTU per hour. Usually after about two hours, however, they can be shut down and a pilot burner is sufficient. The pilot usually is in the range of 3 million BTU/hr. Applying these data to the cupola example for a 10 hour operational period the afterburner would have an average energy input to the cupola system of 6.4 million BTU per hour. This would raise the total energy input of the example to 89,442,115 BTU/hr. The afterburner input adds directly to the sensible heat of the stack gases increasing this percentage of input energy to 53 percent. As will be seen in subsequent sections foundries can no longer allow this energy to be wasted.

AIR POLLUTION CONTROL

Air pollution regulations have caused major difficulties for the foundry industry due to the fact that the emissions from a cupola have inherent properties which make them difficult to control. Not only is the particulate concentration in a cupola's exhaust high but also the particle size of the dust emitted is usually below 10 microns in diameter (see Table 3). To control this within acceptable limits, a pollution control device must be highly efficient over the full range of particle sizes encountered (usually an efficiency of 90% removal is the minimum required, while a removal efficiency of 99% is a common requirement). This efficiency requirement limits the choice of control equipment to three devices; namely, electrostatic precipitators, high energy scrubbers, and baghouses. Relatively simple devices such as

Table 3. PARTICLE SIZE DISTRIBUTION-CUPOLA EMISSIONS (2)

Plant	Cumulative Percent by Weight Diameter in Microns							
	-1	-2	-5	-10	-20	-50	-100	-200
1		30%	50%	65%	82%	90%	99%	
2		64	82	98	99			
3			2	12	34	92	99	99%
4		13	28	45	55	60		
5			54	86	98	99	99	99
6			14	15	15	21	99	
7					19	25	99	
8						99	99	
9		0.6	2	3	8	99	99	
10			4	5.5	7	13.7	75	80
11			11	13	32	53	75	94
12			8	12	17	28	69	69
13			18	25	38	62		
14			17	26	36	53		
15			24	28	23	42		
16			26	30	32	44		
17	0	7	25	32	34	41	56	61
18	0	7	24	41	47	32	69	81

wet caps and cyclones are eliminated. While all three of the applicable pollution control devices mentioned are somewhat hampered by these requirements, the effect is least upon the baghouse. For further information on these devices the reader is referred to Reference 2.

Finally the exhaust gases from a cupola have a carbon monoxide content averaging around 13% . To comply with air pollution codes and to avoid the possibility of explosions in the ductwork, the exhaust gases are afterburned to convert the CO to CO₂. The temperature of the exhaust gas is therefore usually in the range of 2000°F by virtue of the CO combustion. This temperature is, of course, far above what any pollution control device can handle and consequently must be reduced. The two most common methods of accomplishing this are dilution with ambient air and water quenching. Both of these methods, however, present further problems. While diluting with ambient air is an effective method of reducing the exhaust temperature it also significantly increases the gas volume through the system. Since the cost of pollution control equipment is directly related to the air volume to be treated, this method results in a large cost increase. Water quenching systems, while effective to a degree, are not useful in reducing the exit temperature by more than about 500°F. Water quench systems are also frequently incorrectly designed such that not all of the water flashes into steam. This excess water causes damage to the refractory and ductwork. If a baghouse is used even with a well designed water quenching system, cool spots in the baghouse could allow condensation of the water vapor on the bags. This increases the weight on the bag, causes caking, and bag failure. Due to the aforementioned reasons neither system is used exclusively but rather a combination of the two techniques is employed.

With the necessity of employing a highly efficient pollution control device which incorporates a temperature reduction mode the cost of such a system becomes an important consideration to the foundry management. Table 4 shows the economics of air pollution control for the three most prevalent devices. As noted, the cost of any pollution control device is dependent on flow. Using ambient air as a temperature controller can increase the flow by a factor of four or more. The cost of controlling the pollution from a large cupola runs into the millions of dollars. Costs for smaller cupolas are often in the neighborhood of one million dollars. Added to these is the cost of installation which in many cases can be extremely high. This is due to the fact that retrofitting a control device to an existing installation is difficult because of lack of available space or other constraints. Often it becomes necessary to suspend equipment from a superstructure since ground space is not available. Operational and maintenance costs for these devices is also quite high. Venturi scrubbers, besides using large quantities of water which

Table 4. COST EQUATIONS (2)

<u>Equipment Type</u>	<u>Investment Cost Equation</u>	<u>Limits of Observation</u>	<u>Correlation Coefficient</u>	<u>F Ratio</u>	<u>STD Error</u>	<u>Data Points</u>
High energy wet scrubber	$I=49,519+2.84 \text{ gas vol}$	6,000<gas vol<20,000	.82	25	16,000	25
	$I=43,519+8.97 \text{ gas vol}$	20,000<gas vol<92,000	.99	139	29,000	
Low energy wet scrubber	$I=38,744+2.05 \text{ gas vol}$	4,500<gas vol<67,000	.84	55	22,000	34
	$I=55,000+8.95 \text{ vol}$	10,800<gas vol<100,000	.98	321	48,000	19
Fabric filter						

subsequently must be cleaned, have pressure drops frequently in the range of 55 inches of water. Baghouses, while only contributing about 6 inches to the pressure drop, typically have a bag failure rate on the order of one to three bags per day.

All of these factors can be interpreted to mean that air pollution control for a cupola will be expensive in terms of initial investment, operation, and maintenance costs.

While large nationwide foundries may be able to afford the expense of pollution control, the smaller foundries usually have difficulty. Unfortunately, there are too many instances where foundries have spent money for a pollution control system only to discover during performance tests that the equipment is not as efficient as design specifications called for. Although this is sometimes due to careless engineering on the part of the pollution control companies more often it is the nature of the cupola that causes the problems. Underdesigning of systems has been all too common in the past; it has been necessary to scrap the entire system and start anew in some cases.

In summary, numerous problems are associated with air pollution control for the cupola. These problems necessitate expensive solutions. While the need to protect the environmental quality is accepted, the question of how to do so in the most effective and economical way does not have a universal answer.

FUEL COSTS

The second major problem to confront foundries in recent years is the sudden and drastic increase in the cost of coke. The increase in coke prices over the last eight years is graphed in Figure 2. The increases from 1968 to 1973 were caused by two major factors; the Mine Health and Safety Act of 1969 and air pollution regulations limiting the amount of sulfur in coal. These two factors increased the price of the coal available for production to coke thereby increasing the price of coke also. The increase over this period averaged \$5 per ton per year. From 1974 to the present, however, coke prices have jumped enormously, due primarily to the increase in oil prices. The increase over this period averaged \$3.45 per ton per month. This rapid increase appears to be flattening out at the present with the price of coke averaging \$115 to \$120 per ton. The complexities of the world fuel situation make any fuel cost projection untrustworthy. It is extremely unlikely, however, that the prices will go back down or even remain stable for very long.

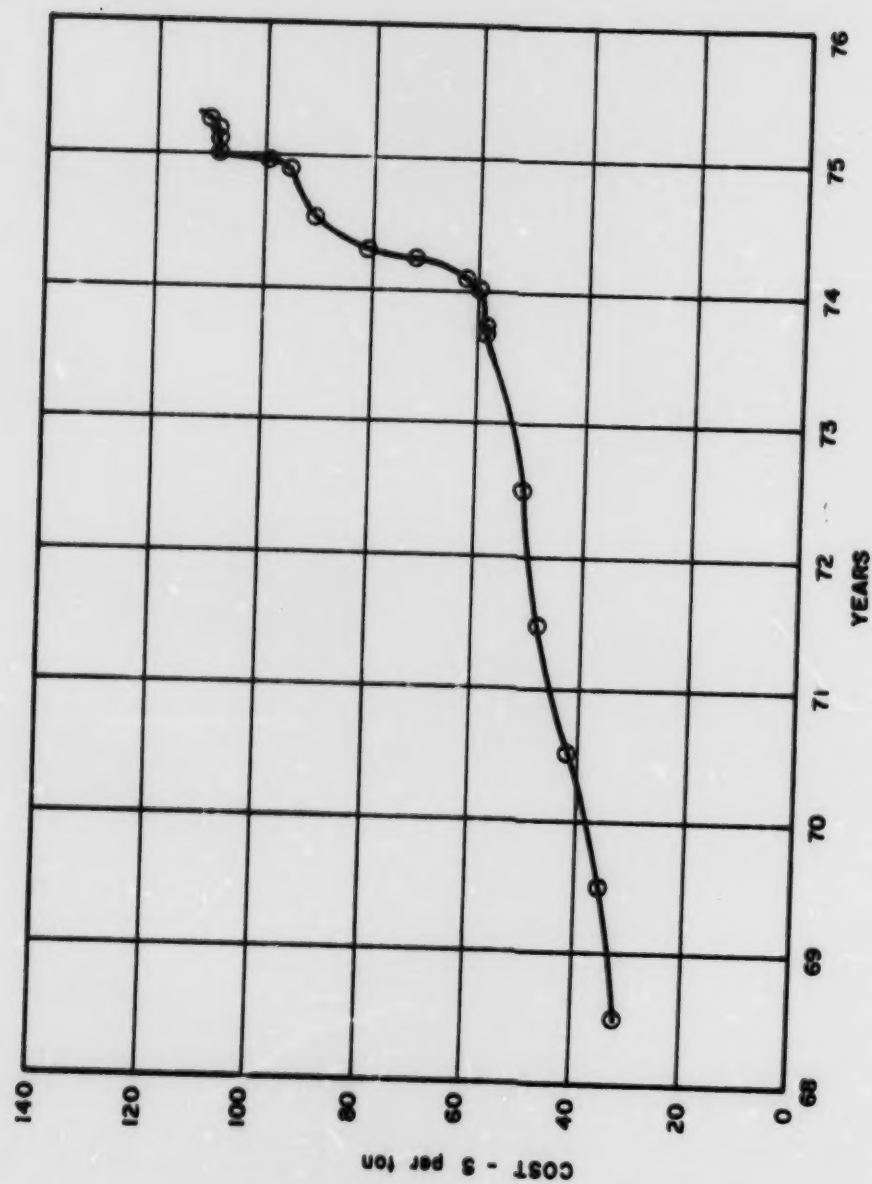


Figure 2. Price of coke 1968-1975.

SECTION V

RECUPERATIVE HOT BLAST

INTRODUCTION

The idea of heating the combustion air of a cupola was known over a hundred years ago. It was not until the 1950's, however, that the idea began to be seriously investigated. It has been shown that a hot blast has many advantages to its use including a decrease in coke consumption and melting losses and an increase in melting rate and iron temperature. It has also been reported that hot blast enables the use of lower carbon contents in the raw materials and allows for less sulfur pickup.

The decrease in coke requirements is the most important aspect of hot blast. The energy in the air substitutes for energy liberated by the coke, thereby reducing the amount of coke required. The relationship between the degree of preheat and coke reduction is shown in Figure 3. For a set tapping temperature a 100°F increase in the blast air makes possible a 0.40% decrease in the charge rate in the amount of coke used.

Example: Given the conditions in Table 5, what fuel cost savings can be realized in a year's time with blast air at 1000°F?

$$\frac{2275 \text{ lbs} \times .004 \text{ lbs of coke reduction}}{\text{charge} \quad 100^\circ\text{F}} \times 930^\circ\text{F}$$

or 84.63 lbs of coke saved per charge or 1.27 tons of coke saved per hour. This is equivalent to 4,762.5 tons of coke per year or a cost savings of \$533,400 per year.

Instead of utilizing the hot blast to decrease coke consumption it can be used to increase the metal tapping temperature. The increase in the metal temperature is approximately 15°F for every 100°F increase in blast air temperature.

Another advantage of the hot blast is to increase the productivity of the cupola by increasing the melting rate. The melting rate of a cupola is a linear function of the coke rate (i.e. percentage of coke in charge). If hot blast is used and since the combustion rate of coke is a constant then by decreasing the amount of coke charged the melting rate will be increased in proportion to the hot blast temperature. A decrease in the coke rate, however, would result in a lower tapping temperature. This can be compensated for, however, by a sufficient increase in air temperature. As can be seen in Figure 4, for a given tapping temperature the melt rate will increase. In such a case the air volume must be the same despite a coke rate decrease since

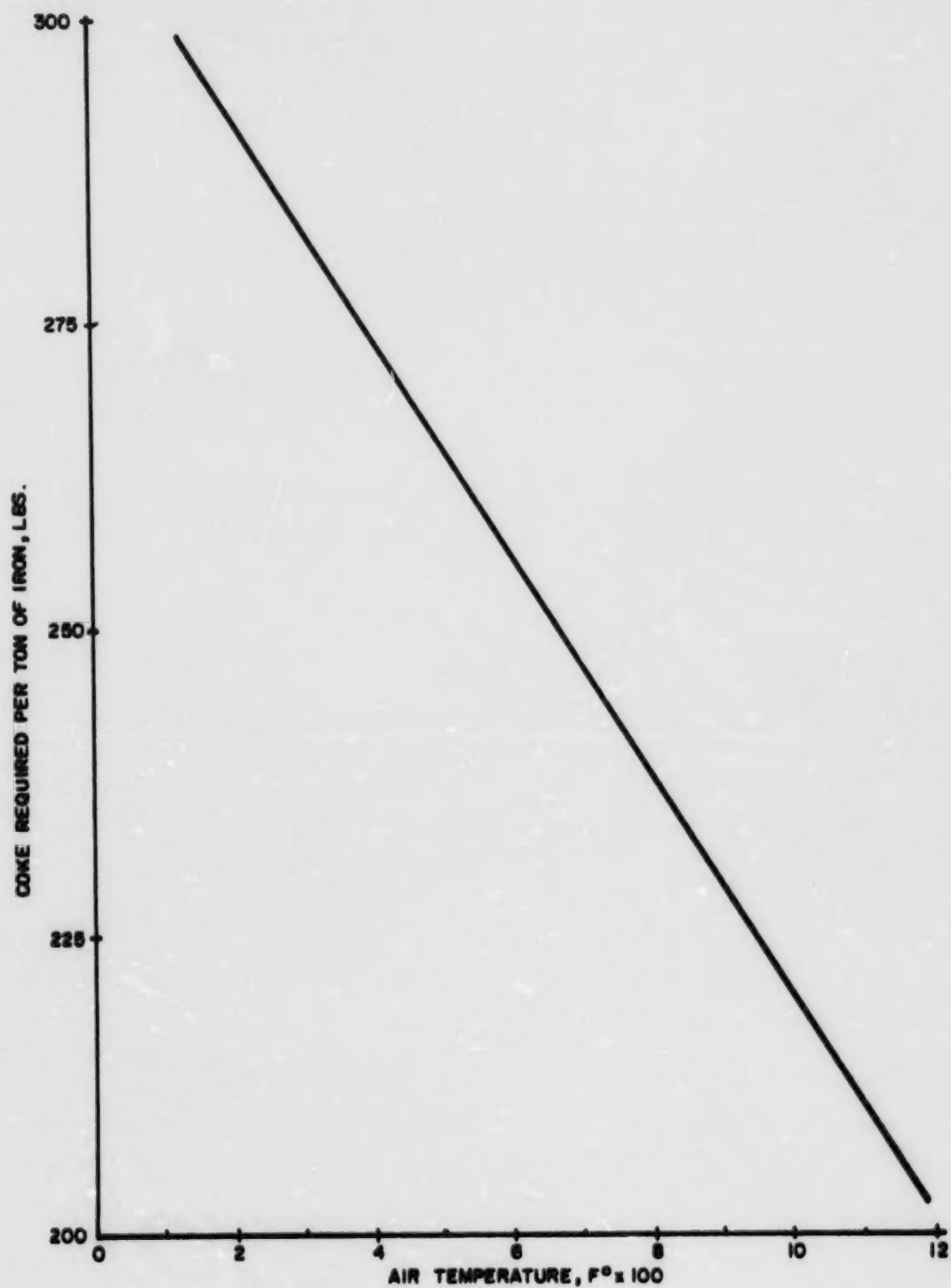


Figure 3. Effect of hot blast on coke consumption. (3)

Table 5. EXAMPLE OPERATING DATA

Melting rate	30 tons per hour
Coke per ton of iron per charge	275 lbs
Ambient temperature	70°F
Hours per day of operation	15
Days per year operation	250
Cost of coke	\$112 per ton

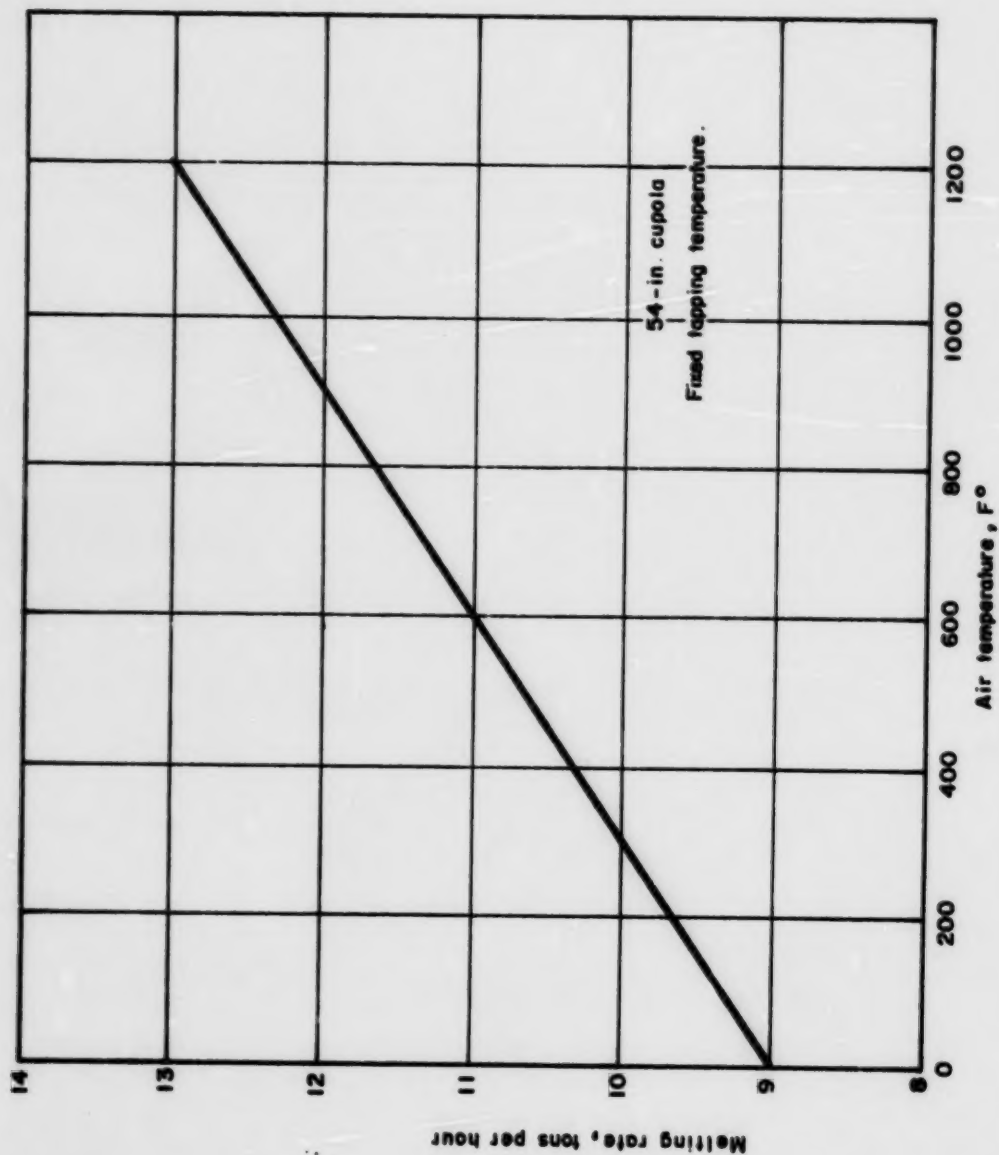


Figure 4. Effect of blast-air temperature on melting rate. (3)

the coke combustion rate must be held constant. If, however, an increase in melting rate is not desired then benefit can be attained by a decrease in the blast rate. This is obvious since the amount of coke is decreased.

Silicon melting loss is also dependent on hot blast temperature. An increase in the air temperature of 100°F appears to decrease silicon loss by 1.5%. This is only a general empirical relationship and can be affected by nature of the charge, slag composition, etc. Sulfur pickup is also decreased by hot blast operation since it is dependent on the amount of coke used and this, of course, is decreased by the use of hot blast. There is also a decrease in the melting loss of iron from hot blast operation. This decrease is approximately 1% when converting from cold to hot blast.

Another major advantage to hot blast is reduced emissions from the cupola. This occurs due to either a reduced coke rate or blast volume. While extensive research has not been performed to correlate these factors with emission rates, limited work has been performed (2) and the following regression equations have been developed:

$$E = 57 - 6.6C + 0.1B \text{ for unlined cupolas} \quad \text{Eq. 1}$$

where E = particulate emissions in lbs per ton of melt

C = metal to coke ratio

B = specific blast rate in SCFM per square foot furnace area

and $E = .05 + .07B$ for acid lined cupolas

While the equations cannot be considered accurate enough for purposes of prediction they do show the relative effect of the variables.

Example: From the conditions listed in Table 5 and with B equal to 300 SCFM/sq.ft. of furnace area, what % reduction in emissions might be expected by converting to hot blast?

As before the amount of coke saved is 84.63 lbs. This increases the metal to coke ratio from 7.3 to 10.5. With a constant B factor then using equation 1 the resulting decrease in emissions is from 38.82 pounds per ton of melt to 17.7 lbs, a decrease in excess of 50%.

There are presently three methods available for preheating the combustion air; namely, gas assisted firing, direct fired heat exchanger and recuperative heat exchanger. The gas assisted cupola is not generally recommended since natural gas while cheap is scarce and also because gas firing in the cupola is inefficient.

The direct fired heat exchanger has been the most popular method used in this country. This was primarily due to two reasons.

First, the heat exchanger necessary could be ordered off the shelf since there were not any severe conditions (such as heavy grain-loading) to contend with. Second, the price of oil made the choice economically feasible. With, however, the rapid increase in fuel oil prices, this can no longer be considered a viable alternative. It appears, therefore, that the recuperative heat exchanger is left as the most likely alternative.

RECUPERATORS

The idea of using the latent and sensible heat of the cupola off-gas to preheat the combustion air has been achieved by recuperative hot blast. A general operational schematic for accomplishing this end is presented in Figure 5. Variations on this schematic will be reviewed later in this section. The cupola off-gas contains energy in two forms. The first is the sensible heat of the air which is directly related to the temperature. The second form is the latent energy which represents the CO content of the gas stream. As can be seen from Figure 5 the cupola off gas is mixed with ambient air infiltrated through the charge door to provide oxygen for combustion and to lower the final temperature by dilution. The gas-air mixture is immediately afterburned to convert the CO to CO₂. Depending on the amount of air added for dilution the temperature of the gas stream exiting the afterburner is usually in the range of 1500 to 2200° F. The gas stream temperature is then further reduced by water quenching. In an effort to reduce the particulate grain loading of the gas stream entering the heat exchanger a low pressure drop cyclone is sometimes included in the system. The cyclone will remove the large particles which may cause some damage to the exchanger and also helps to reduce somewhat the possible fouling of the heat exchanger surfaces. The gas stream then is further cooled by the heat exchanger, followed by another water quench to reduce the temperature to the design limits of the air pollution control device. Since the most prevalent control device for the cupola is a baghouse then the temperature must be reduced to below 500° F. The temperature cannot drop below 212° F or else condensation will occur.

There are four major modifications that can be performed on the basic system. They are above or below charge door takeoff and pollution control before or after afterburning.

The advantage to above charge door takeoff is that the infiltrated air acts to reduce the temperature of the gas stream while supplying air for the CO combustion. There are, however, two main drawbacks to its use. First, if the ID fan is not sized properly dust laden air can escape through the charge door opening. The second and primary disadvantage is that the increase in the amount of air to be cleaned is usually fivefold that blown through the tuyeres. Since pollution control cost is dependent on the air volume to be treated this increase in flow has a significant effect upon both capital and operating cost. One

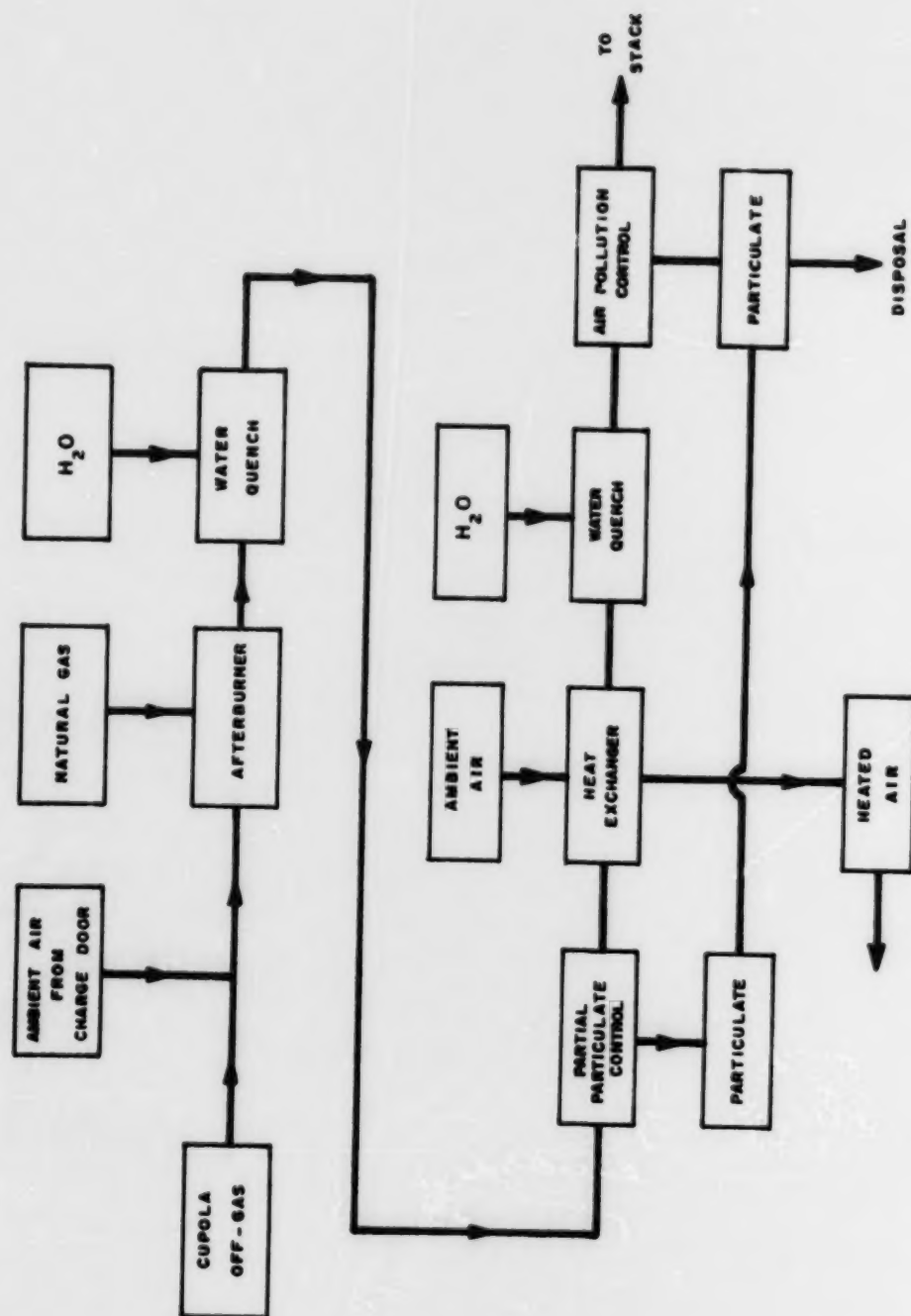


Figure 5. Schematic of heat recovery system

method of reducing the amount of infiltration is to minimize the size of the charge door opening.

The principal advantage to below charge door takeoff is that the amount of air infiltration into the system is much less than that infiltrated with above charge door takeoff. This could, of course, reduce the cost of pollution control and the horsepower requirements of the ID fan. There are basically two different ways of accomplishing below charge door takeoff. Normally, the takeoff point is placed below about five feet of burden. This would reduce the amount of infiltration by about half. Another method is to place about 25 feet of burden over the takeoff point reducing infiltration to approximately zero. This enables particulate removal to occur prior to afterburning. Heat exchangers in such a system can be conventionally designed without special provisions for fouling.

AVAILABLE SYSTEMS

Due to the relative lack of recuperative hot blast installations in this country a survey of manufacturers and installations was made. The purpose was to determine if available systems were reliable. The following firms were identified as manufacturers having installations in this country: Escher, Inc.; McQuay Perflex; Thermal Transfer; American Schack and Combustion Equipment Associates. Other firms which have sold systems to foundries which are not yet installed included F.E.C.O.R. and Q-Dot. Finally, some engineering firms having experience with the cupola have designed systems but have yet to market them extensively. An example of this type of firm is Smith Environmental Corporation. An attempt was made to visit representative foundries which had all of the various manufacturer's equipment installed as well as receiving engineering descriptions of those systems in the planning stages. Permission to visit the Allegheny Ludlum Steel Corporation which had the only American Schack equipment installed in this country was not granted for proprietary reasons involving their duplexing process. Also, Q-Dot Incorporated did not furnish any engineering information on their equipment. All other requests for vital information and visits were granted.

Escher Incorporated

Escher's equipment accounts for 17 out of the 20 known recuperative heat exchangers installed in this country. They supply both the conventional "above charge door" takeoff system as well as "below charge door" systems. They have both types of systems in use in this country. A listing of a representative sample of Escher's pricing can be found in Table 6. These costs are for recuperators accepting 100% of the flue gases and preheating the combustion air to 950°F. These costs do not represent installation costs which can vary appreciably depending on the foundry arrangement.

The Escher heat exchanger is a counterflow type utilizing

TABLE 6. RECUPERATIVE HOT BLAST SYSTEMS

No.	Installation	Location	Start-up	Manufacturer	Capacity T/H
1	Glamorgan Pipe Foundry	Lynchburg, Va.	1969	Escher	45
2	Chevrolet Grey Iron Foundry (1) Chevrolet Grey	Saginaw, Michigan	1970	Escher	35
3	Iron Foundry (2) Chevrolet Nodular	Saginaw, Michigan	1970	Escher	35
4	Iron Foundry Chevrolet Grey	Saginaw, Michigan	1970	Escher	65
5	Iron Foundry (3)	Saginaw, Michigan	1970	Escher	65
6	Auto Spec. Mf. Co. Chevrolet Grey	Benton Harbor, Michigan	1970	Escher	45
7	Iron Foundry (4) Chevrolet Grey	Saginaw, Michigan	1971	Escher	65
8	Iron Foundry (5)	Saginaw, Michigan	1971	Escher	65
9	U.S. Pipe & Foundry	Chattanooga, Tennessee	1971	Escher	40
10	U.S. Pipe & Foundry	Birmingham, Alabama	1972	Escher	70
11	U.S. Pipe & Foundry	Bessemer, Alabama	1972	Escher	70
12	U.S. Pipe & Foundry	Burlington, New Jersey	1973	Escher	70
13	East Jordan Iron Works	East Jordan, Michigan	1973	Escher	55
14	Kelsey-Hayes Gunitite	Rockford, Illinois	1974	Escher	40
15	U.S. Pipe & Foundry	Anniston, Alabama	1975	Escher	25
16	Chrysler Corp.	Fostoria, Ohio	1975	Escher	32
17	Eljer Plumbing Ware	Salem, Ohio	1976	Escher	15
18	Berlin-Chapman Foundry	Berlin, Wisconsin	1971	McQuay-Perfex	11
19	Allegheny-Ludlum Steel Corp.	Pittsburgh, Penn.	1966	American Schack N.A.	
20	U.S. Pipe & Foundry	Union City, Cal.	1960	Thermal Transfer	40

Table 7. . ESCHER RECUPERATOR PRICE LIST
FOR BLAST PREHEAT OF 950°F

<u>Blast Volume</u> <u>SCFM</u>	<u>Melt Rate</u> <u>TPH</u>	<u>Cost</u>
2000	5	\$ 35,000
4000	10	49,400
6000	15	68,500
8000	20	86,800
13,500	32	132,900
19,000	40	171,500

a radiation and convection section. The inner shell is surrounded by an outer shell and a small portion of the blast air flows between them. The major portion of the blast air flows through the hollow fins located in the gas stream. The preheated blast air is collected in a circular chamber prior to entry into the blast piping leading to the cupola (see Figure 6).

To accommodate the expansion of the blast heater a flue gland is provided between the heater and the flue gas inlet duct. A flexible hot blast duct is also provided.

The hot blast temperature is maintained by controlling the inlet flue gas temperature or by controlling the flue gas flow through the heat exchanger.

Escher's equipment can be used in series or parallel applications and can be followed by any type of pollution control device.

The materials of construction for the Escher heat exchanger are a combination of stainless and mild steel to minimize cost.

One of the major points to be considered about any recuperative device is how it is affected by the particulate grain loading. The Escher units are self-cleaning devices by virtue of the vertical arrangement, tube design and gas velocity through the exchanger. Particulate also has an effect upon the allowable incoming temperature since ash from the cupola can start to form sintered deposits at 1800°F.

Plant Survey-

U.S. Pipe and Foundry, Burlington, N.J. Plant data can be found in Table 8 and a schematic of the system in Figure 7. As can be seen the parallel Escher units deliver approximately 25,000 SCFM at 950°F to the cupolas. The hot blast allows for a 10 to 1 metal to coke ratio. The flue gas from the afterburners (modulating control) goes to three spray nozzles which are temperature controlled to reduce the temperature to below 1500°F. The heat exchangers are one stage units with the exiting gas going directly to the final quench and then to the baghouses.

When the plant visit was made the units had been in operation for over a year with no sign of deterioration. A maintenance check during the foundry's vacation period showed no sign of deterioration in the structure or the material of the exchanger. Some rubber seatings did have to be replaced but according to the foundry personnel this was a quick inexpensive procedure.

Plant Survey-

There are five Escher units installed at the Chevolet Foundry, GMC, Saginaw, Michigan. All units are of the "below charge

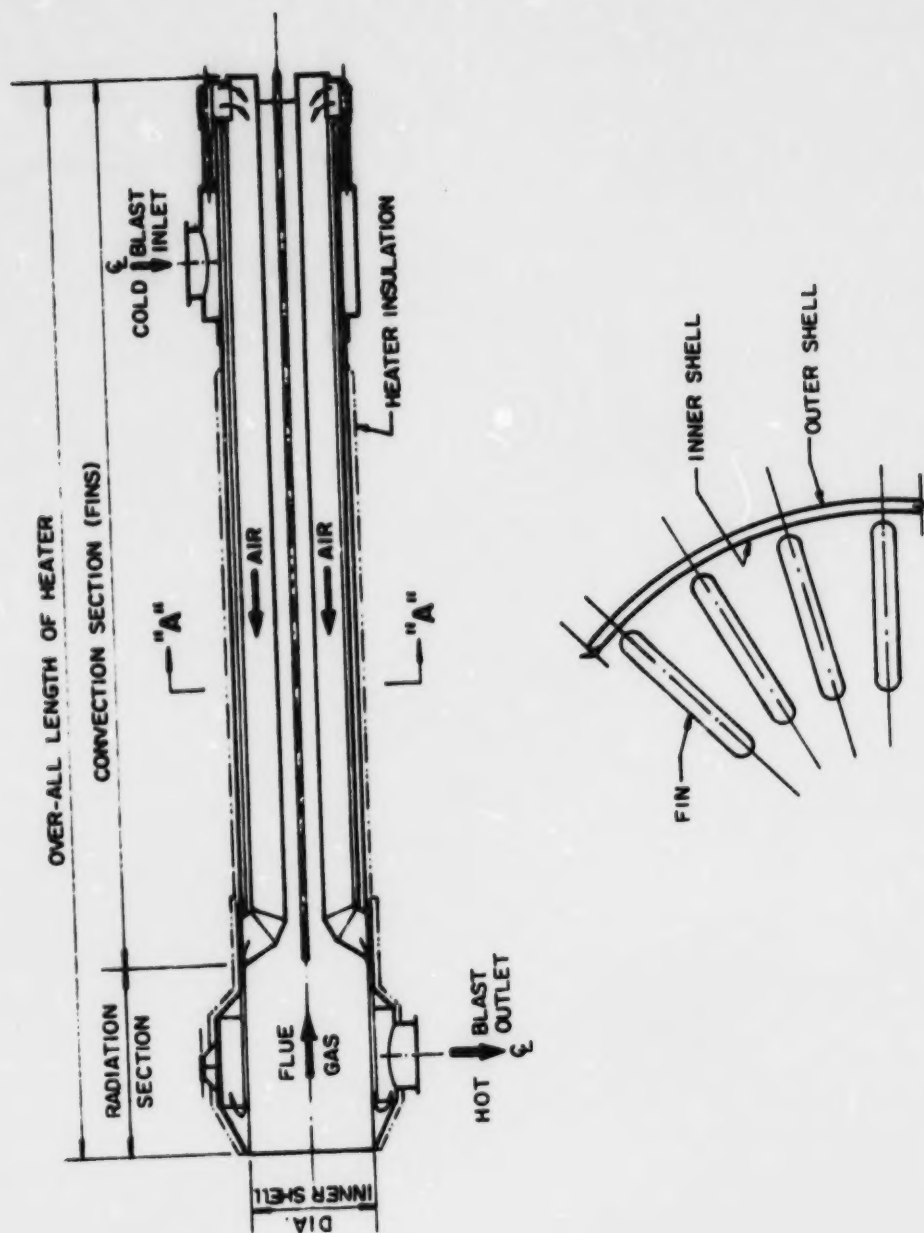


Figure 6 . Escher recuperative blast heater .

Courtesy of ESCHER, INC.

**Table 8. PLANT DATA, U.S. PIPE AND FOUNDRY COMPANY,
BURLINGTON, N.J.**

Type of Product	Ductile
Melt Rate TPH	58
Blast Rate SCFM	25,000
Blast Temperature °F	950
Oxygen Enrichment %	2-4
Metal to Coke Ratio	10/1
Exhaust Flow Rate SCFM	67,800
Exhaust Temperature °F	1,800
Charge Door Opening Ft ²	25
Afterburner 10 ⁶ BTU/hr	30
CO Content %	13

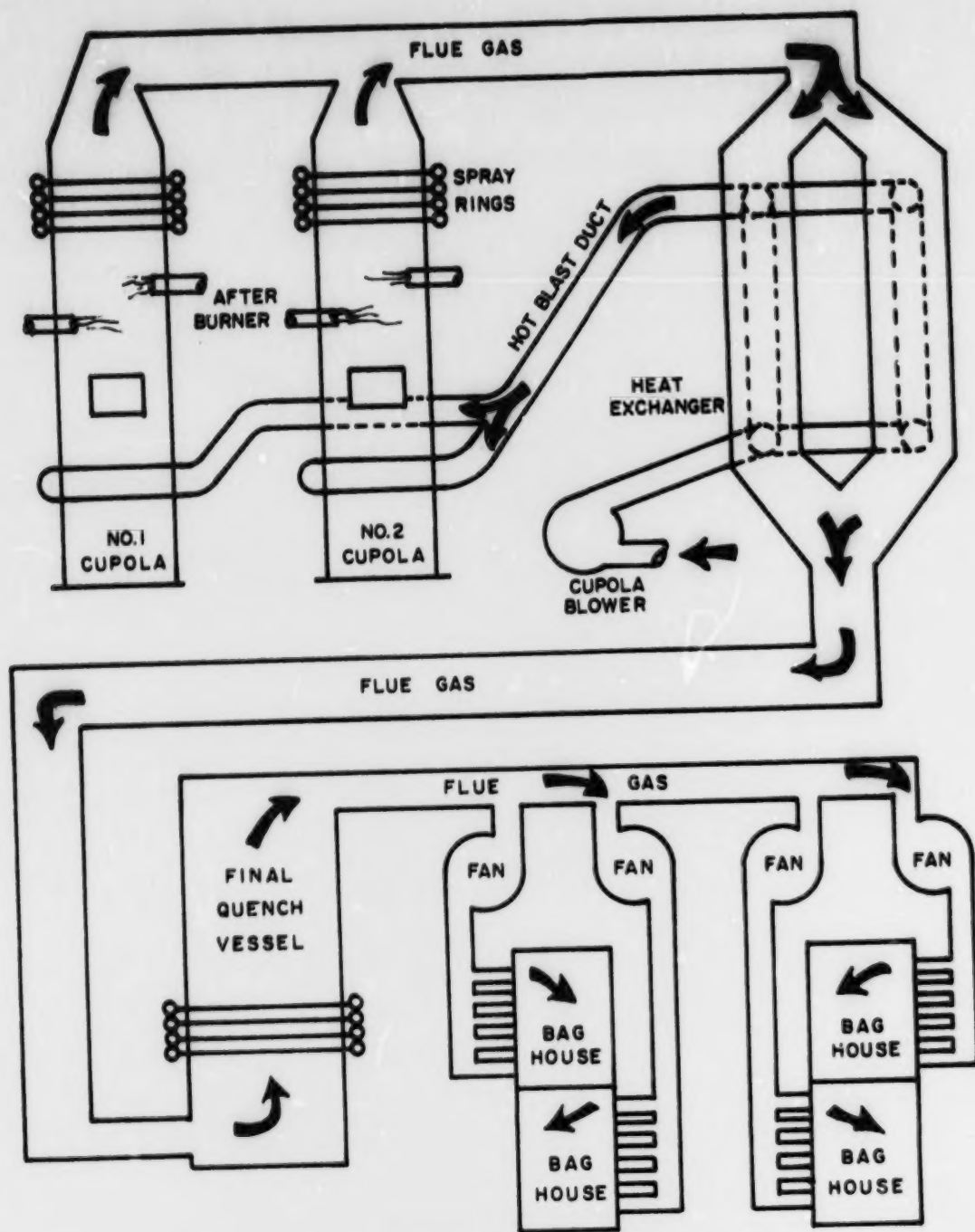


Figure 7 . Plant schematic , U.S.Pipe & Foundry Co., Burlington , New Jersey

door" takeoff type. The first unit was installed in the late sixties and performance after some initial difficulties has proved satisfactory to the foundry management. There appears to have been some trouble with the first installation primarily due to excessive flue gas temperatures entering the heat exchanger. After this was corrected the unit performed as designed and there have been no problems with any of the other units. Routine maintenance procedures have eliminated the need for any extensive overhaul.

An innovative approach to energy conservation had also been taken by GM engineers by using the energy from one of their cupolas for plant heating. Approximately seventy million BTU per hour are made use of in this manner contributing to substantial savings in fuel cost. Plant data for the two different size cupolas can be found in Table 9.

The arrangement of the system can be found in Figure 8. Due to the size of the cupolas, however, it was necessary to install a by-pass cooler. It is the heat from the by-pass and evaporative heat exchanger that is being used to supply the plant heating.

An interesting point to note concerning the operation of these cupolas is the high coke rate. According to the foundry engineers this is due to the fact that there is no pig iron in the charge material. As a result the carbon pickup must come from the coke and thus the high coke rate. It is also interesting to note that the blast temperature is given in a 400°F range. Again, metallurgical considerations are the prime reasons for a given blast temperature and the cupola operators have greater control of the final product by manipulating the blast temperature. All of the Escher systems are capable of operating over a blast temperature range of approximately 400°F. The degree to which this range is used depends upon the particular foundry operating practice.

McQuay Perfex Inc.

Perfex's recuperative heat exchanger was designed to provide hot blast to a cupola with a minimum of maintenance. The heat exchanger consists of 14 tubes, 92 inches long and 12 inches in diameter. The tubes are equipped with both internal and external fins to aid in heat transfer. The internal fins through which the dust laden off-gas from the cupola is forced are designed to present an inclined spiral shape to the gas flow. These fins are cleaned daily by an air lance and can be used during operation. A representation of the inner fin arrangement can be seen in Figure 9.

McQuay Perfex presently offers two sizes of heat exchangers. Data on the size of the units can be found in Figure 10 and data on pressure drops, flow and blast temperatures is presented in Figure 11. The cost of the two units is \$30,000 for the smaller unit and \$41,000 for the larger.

Table 9. PLANT DATA, CHEVOLET FOUNDRY, GMC,
SAGINAW, MICHIGAN

Type of Product	Grey
Melt Rate TPH	35
Blast Rate SCFM	18,000
Blast Temperature °F	600-1000
Oxygen Enrichment %	Max. 4
Coke Rate %	18
Exhaust Flow Rate SCFM	60,000
Exhaust Temperature °F	2000
Afterburner 10 ⁶ BTU/hr	17
CO Content %	18
I.D. Fan HP	350

Type of Product	Grey
Melt Rate	55
Blast Rate	25,000
Blast Temperature °F	600-1000
Oxygen Enrichment %	Max. 4
Coke Rate %	18
Exhaust Flowrate SCFM	70,000
Exhaust Temperature °F	2000
Afterburner 10 ⁶ BTU/hr	17
CO Content	18-20
I.D. Fan HP	1250

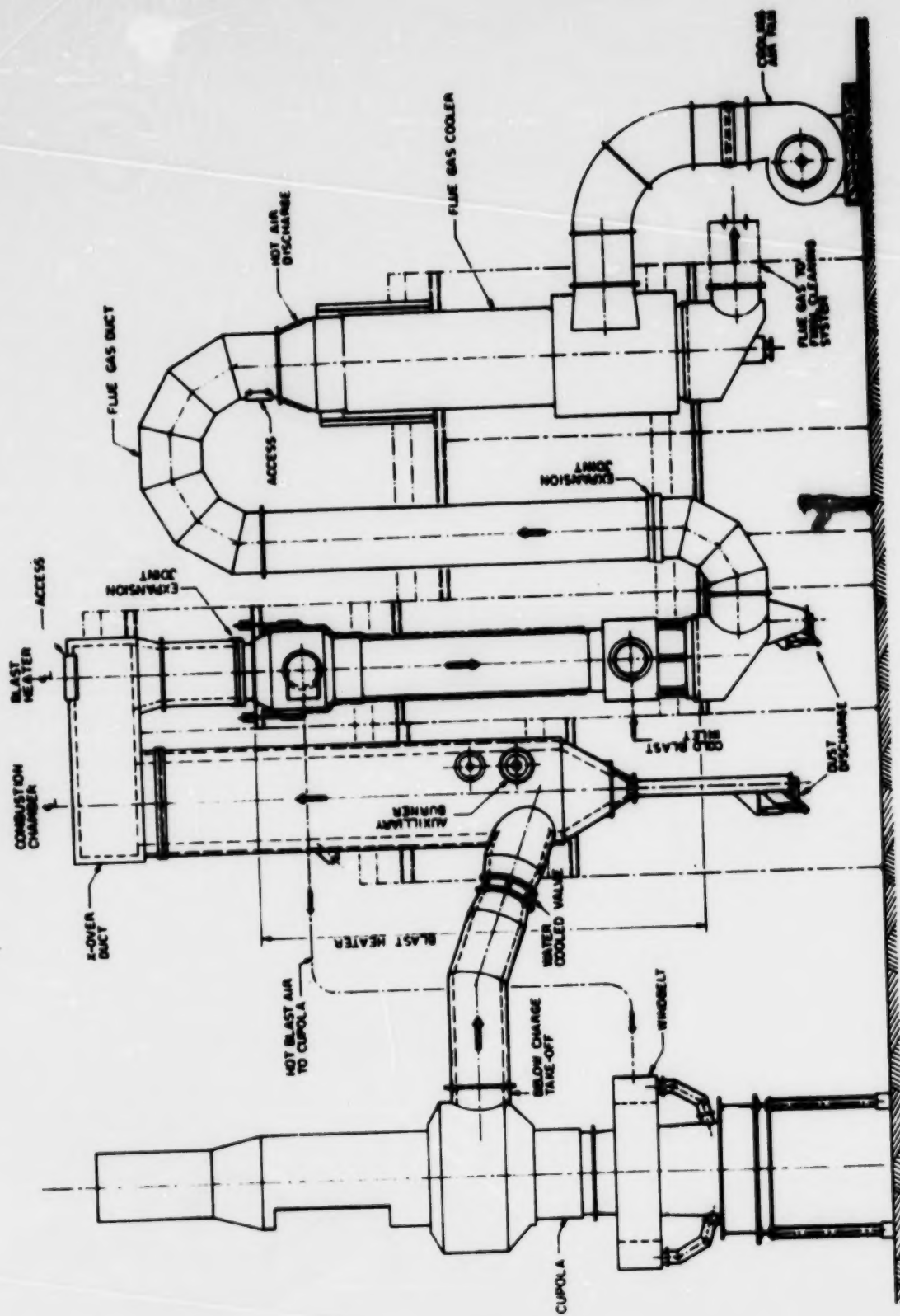


Figure 8 - Typical Escher Recuperative Arrangement

Courtesy of Escher, Inc.

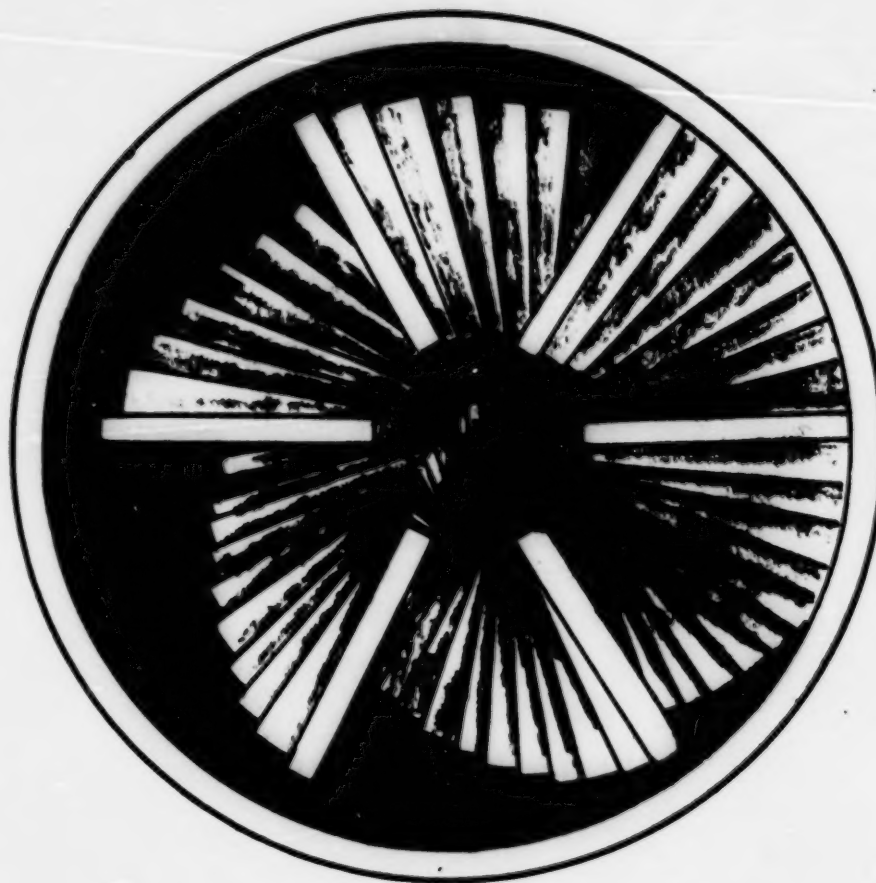


Figure 9. Inner tube fin arrangement.

Mod.	Dim.	A	B	C	D	E	F	G	H	J	K
C2		2'-0"	3'-0"	6"	1'-0"	19'-0"	5'-0"	3'-0"	13'-0"	3'-0"	73"
P2		2'-6"	4'-0"	6"	1'-0"	23'-0"	4'-9"	3'-6"	14'-0"	4'-0"	85"

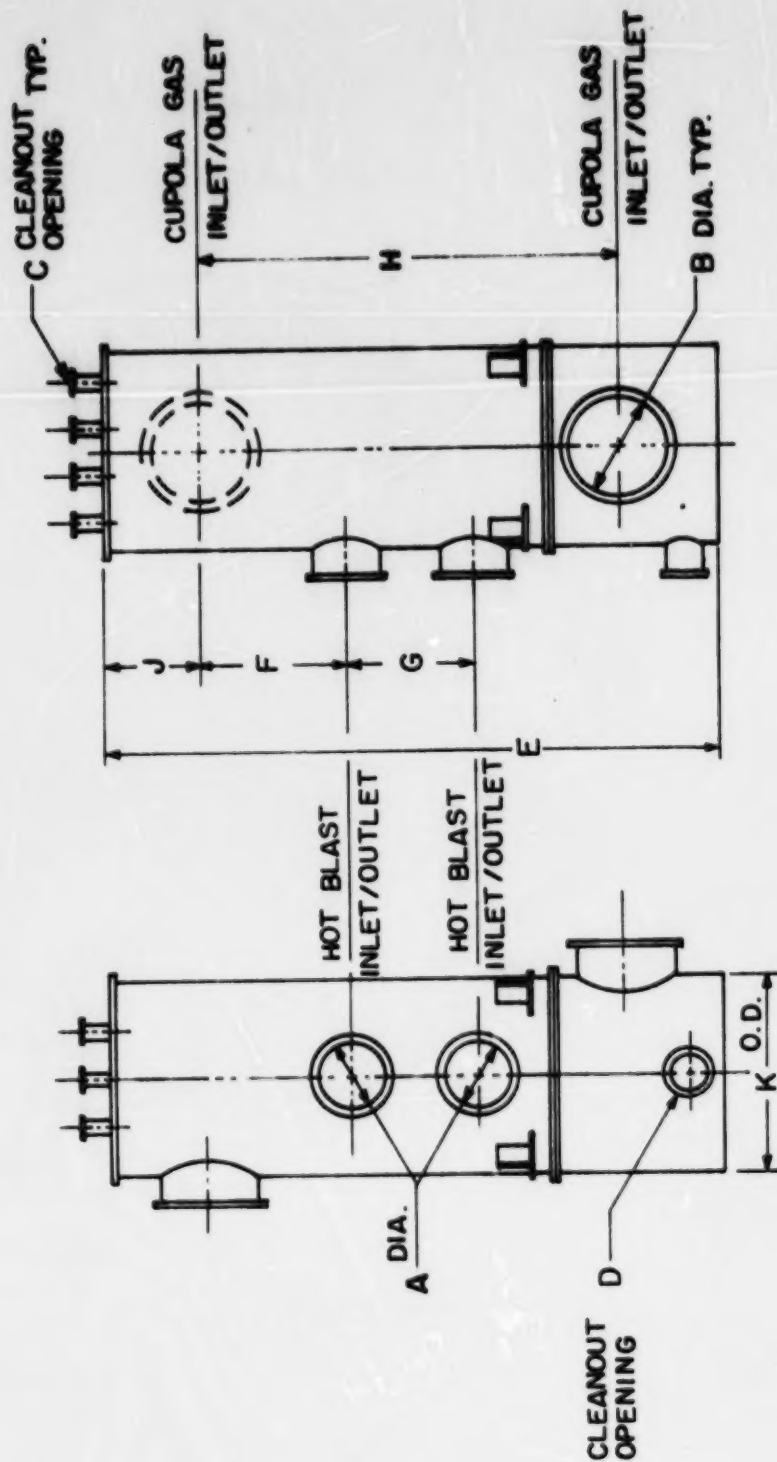
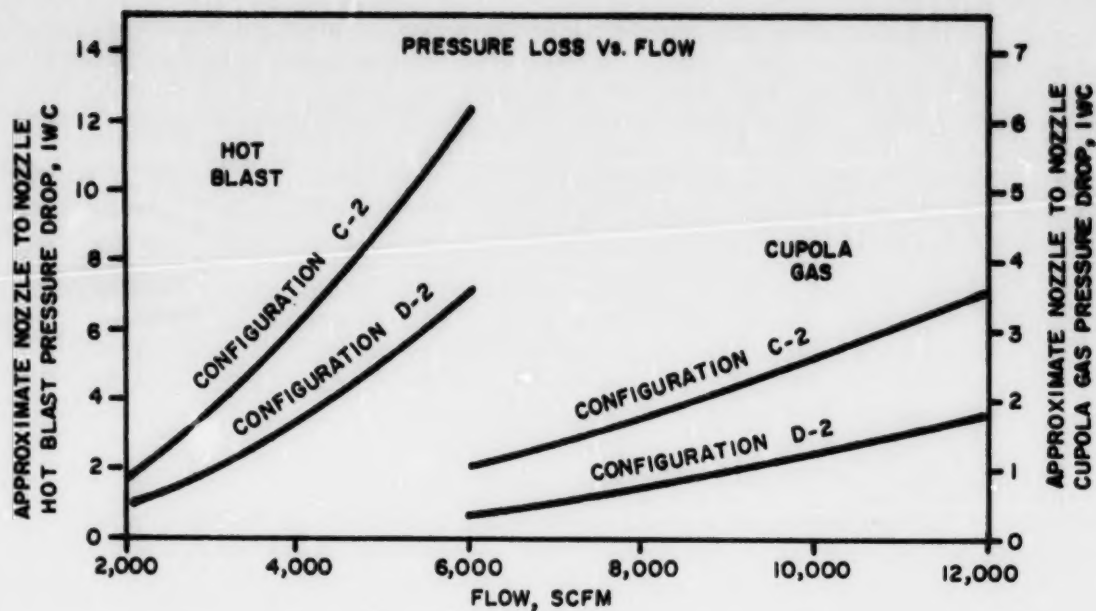


Figure 10. Dimensions, McQuay-Perfex recuperator .



ASSUMPTIONS:

1. Hot blast inlet temp. = 0°F
2. Cupola gas inlet temp. = 1200°F
3. Unit 85% clean
4. Flow as shown schematically

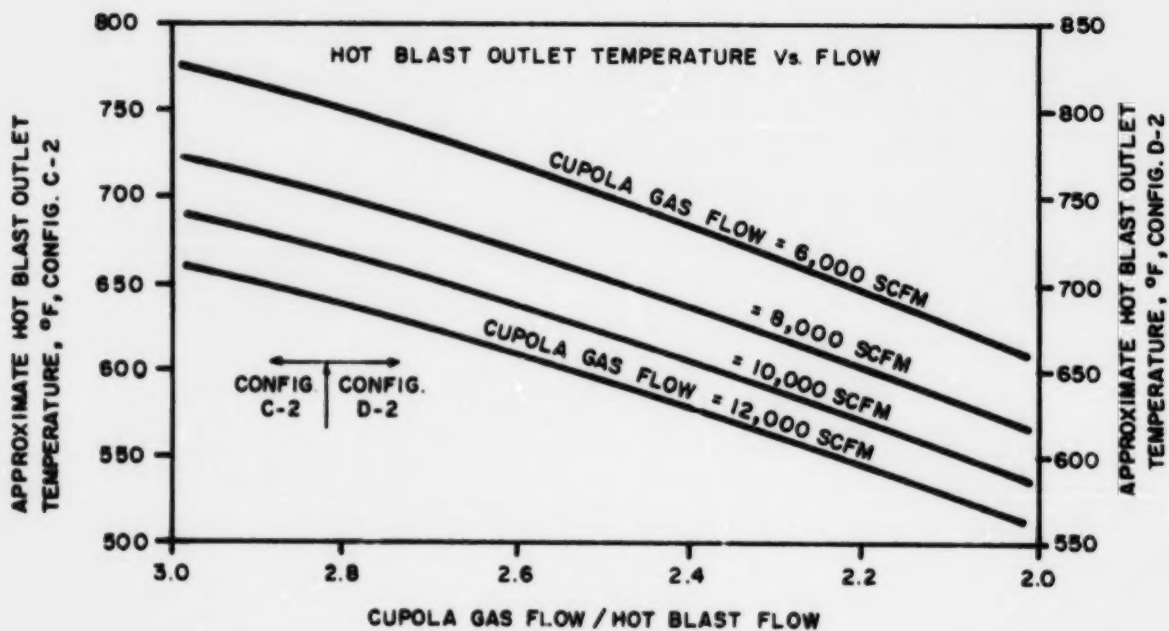
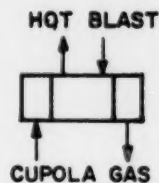


Figure 11. Pressure, temperature relationship for McQuay - Perflex recuperator.

Plant Survey-

Berlin-Chapman Foundry, Berlin, Wisconsin cupola operating data can be found in Table 10. The recuperative heat exchanger has been in operation for four years with no sign of deterioration. The total cost of cleaning the heat exchanger for a year's period is estimated at \$3,000. A schematic of the system layout is depicted in Figure 12. The pollution control device is a scrubber type system with the trade name "Air Sweep" and it is marketed by McQuay Perfex Inc.

The heat exchanger is composed primarily of carbon steel materials with the exception of the fins which are stainless steel. The hot blast operation has increased the melting rate of the cupola by 25 percent. Due to the increase in melting rate no change in the coke percentage was achieved.

Thermal Transfer Corporation

Thermal Transfer Corporation has one installation of a heat recovery system on a cupola. The installed unit has been in operation for over fifteen years and the design is a conventional shell and tube type with no inherent design features to overcome fouling.

Plant Survey-

U.S. Pipe and Foundry, Union City, California cupola operating data is presented in Table 11. The main disadvantage of this system is that it is not self-cleaning. A shot-cleaning system is employed and there were numerous start-up problems associated with it. Subsequent problems with the system would indicate that such a design should not be used in the future. U.S. Pipe and Foundry and Thermal Transfer should be commended, however, for being farsighted enough to install a heat recovery unit and make it work when they did.

American Schack Company Inc.

At present American Schack has one recuperator installation in this country at Allegheny Ludlum Steel Corporation. Due to the proprietary nature of the process permission to visit the facility was not granted.

The installed recuperator is a below charge door type which has the take-off point below approximately 15 feet of charge material. The off-gas is then water quenched and cleaned by use of a scrubber type system marketed under the name of the "Theisen Disintegrator" by American Schack.

Table 10. PLANT DATA, BERLIN-CHAPMAN FOUNDRY,
BERLIN, WISCONSIN

Type of Product	Grey, Ductile
Type of Cupola	#5, Whiting
Melt Rate TPH	11
Blast Rate SCFM	4,000
Blast Temperature °F	700
Coke Rate %	13.3
Exhaust Flow Rate SCFM	13,462
Exhaust Temperature °F	1200
Pressure Drop Across Recuperator (Hot Side) Inches of Water	3.4
Pressure Drop Across Recuperator (Cold Side) Inches of Water	4.0

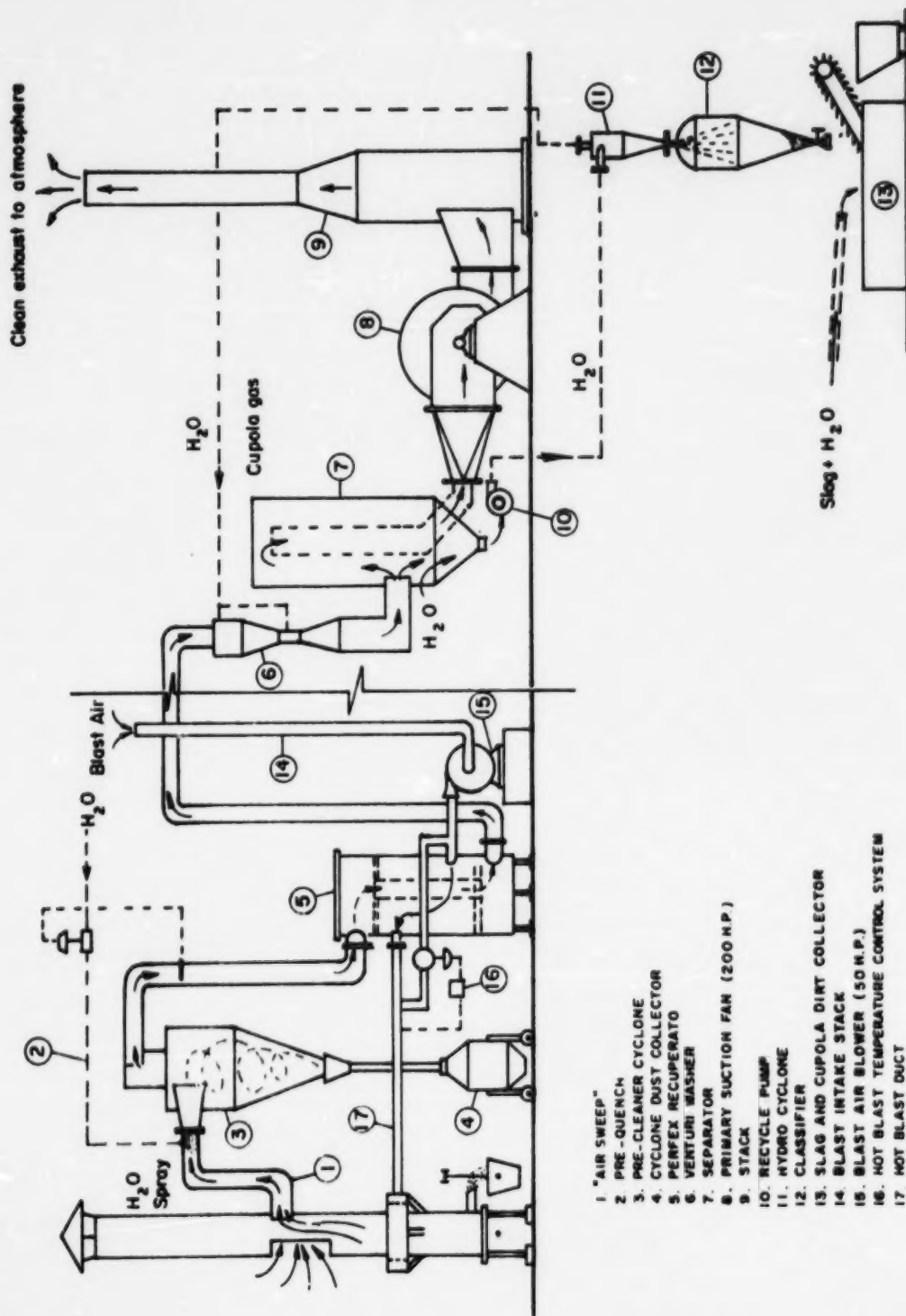


Figure 12. Recuperator schematic, Berlin-Chapman Foundry

Courtesy of McQuay-Perfex, Inc.

Table 11. U.S. PIPE AND FOUNDRY, UNION CITY,
CALIFORNIA

Melt Rate TPH	30-40
Blast Rate SCFM	11,000-13,000
Blast Temperature °F	600-900
Coke Rate	8.2
Exhaust Flow Rate SCFM	50,000
Exhaust Temperature °F	1400
Afterburner 10 ⁶ BTU/hr	3
CO Content %	Not available

The cleaned gas has a heating value of approximately 50 BTU/SCF. All of the gas is combusted by an ignition burner of 500,000 BTU/hr capacity. This burner is in constant operation to insure complete oxidation of the CO. Exhaust gas is continuously analyzed for CO, H₂ and O₂ both for combustion and safety considerations. In the case of high O₂ or H₂ concentrations a valve is automatically activated exhausting the cleaned gas to the atmosphere.

The recuperator at Allegheny Ludlum has been in use since 1966 and has not needed any major repair work outside of tube replacement. The major benefit to pollution control prior to combustion is that the heat exchanger does not need any special design considerations to overcome fouling.

F.E.C.O.R. Industries

This system is similar in concept to American Schack in that the exhaust gas is cleaned prior to combustion. A schematic of the F.E.C.O.R. system is presented in Figure 13. As can be seen the gas take-off point is below the charge door. There is approximately twenty feet of charge material above the take-off point. The exhaust gas from the cupola is water quenched to about 160°F prior to entering the venturi scrubber. The gases are monitored for O₂, H₂ and CO and a sophisticated control system is used to switch the gas stream into a by-pass mode if so indicated. The by-pass system includes an afterburner to combust the CO. During by-pass mode operation or during start up and shut down, hot blast operation can be accomplished by firing the heat exchanger with oil or natural gas.

The F.E.C.O.R. system is being installed at the Central Foundry Division of General Motors Corporation in Defiance, Ohio. Design data for the proposed facility can be found in Table 12. The F.E.C.O.R. system is not suited for retrofitting to existing installations due to the necessity of keeping a twenty foot charge burden above the take-off point. The cupola must be designed to hold the charge and constructed to handle the extra weight. Also, the top of the cupola must be able to be sealed during melt down after the charge is dropped to prevent oxygen infiltration.

Combustion Equipment Associates, Inc.

The original intent of the EPA contract under which this report was written was to conduct an extensive testing and evaluation program on the integrated energy recovery-air pollution control system purchased by the Flynn and Emrich Company from Combustion Equipment

Associates (C.E.A.). Reference 4 contains a detailed description of the system including projections of the operation cost savings. The system, however, was never made operational due to severe design deficiencies centering on undersizing of the ducts and fan. Reference 5 discusses these shortcomings in detail and describes the instrumentation selected for the test program. The computer model which was to be verified with the test data and then used to extend the results to other operations is also described.

The novel feature of the F&E installation was the dry media heat exchanger which employed ceramic balls as the heat transfer media. Projections indicated that such a device would allow for operation at a higher temperature and would result in a more compact unit than other types of heat exchangers. Since the unit was never made operational, no definite conclusions can be drawn. However, the basic concept is still valid and is worthy of further investigation. CEA is no longer marketing the system. Mr. Paul Vandenhoeck, holder of the patent on the heat exchanger, is now with Richmond Tech-Air Corporation, Mountainside, New Jersey.

ECONOMIC EVALUATION OF RECUPERATIVE HOT BLAST

There are two primary economic advantages to recuperative hot blast operation. First, with hot blast operation a coke reduction of 20% can be experienced for an average cupola. Second, with recuperation, the cost of hot blast operation after initial capital cost is limited to the fan amperage required to overcome the increased pressure drop and minor maintenance. To determine if the capital expenditure for the recuperator is justified a number of points must be considered. First, it must be determined that hot blast operation will not adversely interfere with the product composition and that the space requirements of the recuperator can be met. If the charge material composition remains the same then a decrease in coke can be expected. A life cycle cost analysis of the recuperator should then be performed and the results compared with the savings expected from the coke reduction. A decision should then be made consistent with the company's fiscal policy.

The plants visited during this investigation did not include any which were retrofitted with recuperators but rather the recuperators were constructed to be fitted to the cupolas from initial start up of the plant. Data on fuel savings were gathered from other investigators as was previously mentioned but are not inconsistent with data given in this report. Those plants which have high coke percentages usually use what is considered poor charge materials. The excess coke was utilized to increase the carbon content of the product.

A point to keep in mind is that when going out for bid the contractor should guarantee not only the equipment but the coke percentage and product composition. This is a practice common of the more reliable manufacturers.

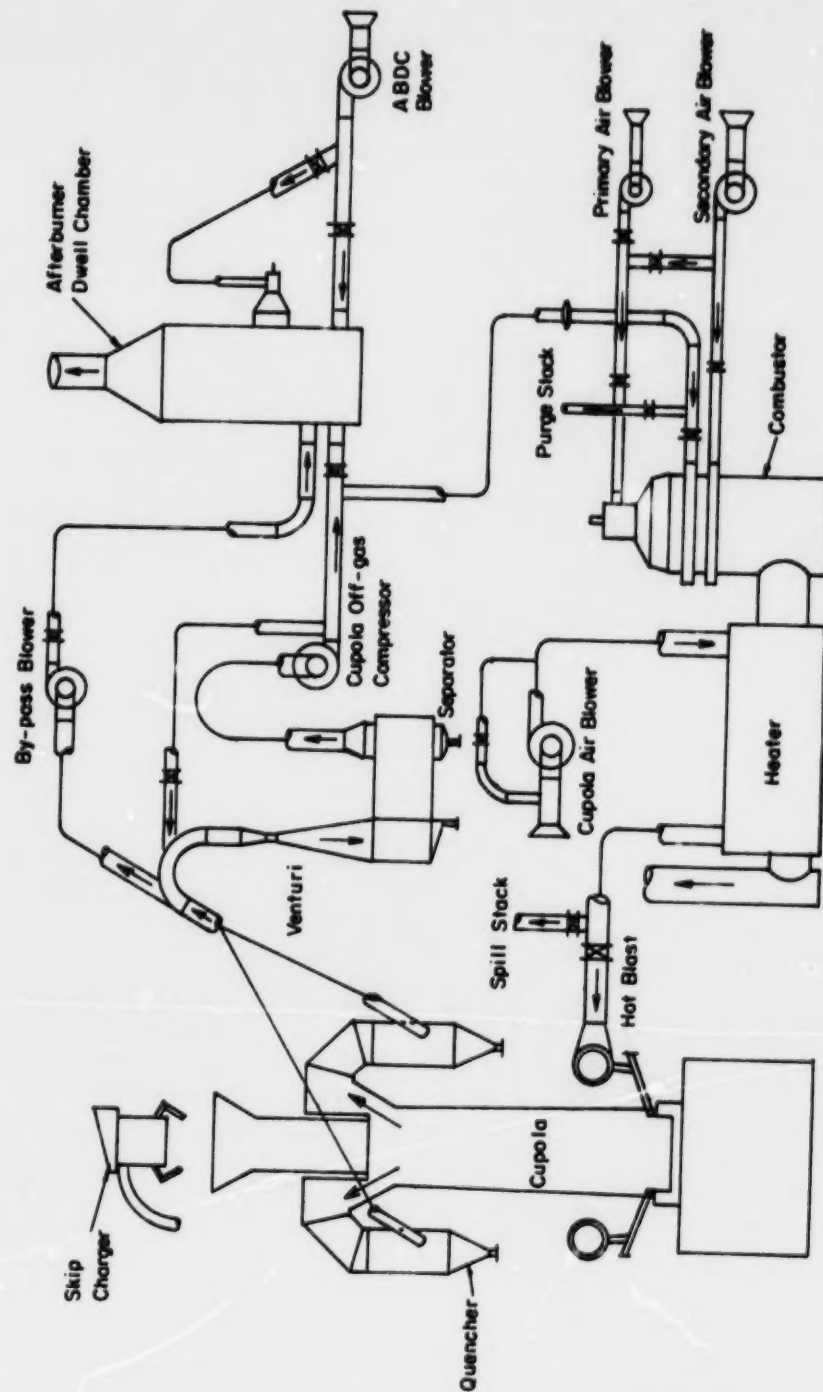


Figure 13. FECOR Recuperative system.

Courtesy of FECOR INC.

Table 12. DESIGN DATA, CENTRAL FOUNDRY DIVISION

Type of Cupola	No. 4 Whiting
Type of Product	Grey Iron
Melt Rate TPH	63
Metal Temperature °F	2700
Blast Rate SCFM	30,000
Blast Temperature °F	1200
Temperature to Exchanger °F	4600
Oxygen Enrichment %	2-4
CO Content %	18
Minimum Combustion Content of Exhaust Gas BTU/C.F.	60

SECTION VI

OXYGEN ENRICHMENT OF THE CUPOLA WIND

INTRODUCTION

It has been well known for many years that a controlled addition of pure oxygen to enrich the combustion air in gray iron cupolas can have marked beneficial effects upon the performance of this type of furnace.

For the most part oxygen enrichment has been used on an intermittent basis, generally as a corrective measure. It has been found that the time required to restore normal operating temperature in the cupola after unscheduled interruptions can be reduced by 50% or more with oxygen enrichment.(6) The reason for its effectiveness is that an immediate response is obtainable by using 2 to 4% oxygen enrichment levels without adjustment of the blast velocity.(7) This is due to the fact that a faster combustion reaction is brought about by the concentration of oxygen in the blast, with a corresponding decrease in the nitrogen volume. The nitrogen, acting as an inert ingredient in the blast air, tends to cool the flame and carries combustion heat up the stack. Some of this heat is recovered by the incoming charge, but adding more air causes excessive gas velocity through the coke bed, increasing heat loss and giving rise to channeling along the cupola wall with attendant radiation loss and refractory erosion.

Significant raw material savings have been achieved by reducing the wind rate and adding 2 to 3% oxygen continuously.(6) This procedure produces higher flame temperature and a shorter oxidation zone in the oxygen enriched coke bed, providing a melting atmosphere richer in CO at any given metal-coke ratio without sensible heat losses from nitrogen. The important result of this is to melt iron with 15 to 20% less coke than normal without the metal oxidation which would have occurred with conventional combustion air rates.(7) The lower velocity of gaseous materials for each ton of iron melted allows more even burning of the coke and reduces the danger of channeling.

It should also be noted that the stack emission load is reduced in proportion to the drop in blast volume. The extent of this reduction is dependent upon the amount of air influx at the charging door. An air pollution control system that is marginal in its ability to treat the volume of stack gas produced by conventional practice will improve in performance with the reduced volume provided with oxygen enrichment.

Additional production benefits have been derived by the use of oxygen enrichment during cupola startups. Case studies have shown

that the time from initiation of the furnace blast to the first tap have been reduced by as much as 58%, with increases in metal first-tap temperatures of approximately 50°F. (6) In this type of application the cost of oxygen consumed is more than offset by savings realized through the reduction or elimination of iron normally pigged, and a reduction in misruns and faulty castings produced due to "cold" metal defects after initial start-ups.

The use of oxygen enrichment on anything other than an intermittent basis has until fairly recently been limited to the larger foundries due to the relatively high cost of liquid oxygen as compared with the cupola raw material savings that could be realized by the application of enrichment techniques. The unit cost of oxygen is primarily dependent on consumption levels as illustrated in Figure 14.

Spiralling raw materials costs in the past five or six years have, however, forced a change in this philosophy. Improvements in air separation techniques and plant design have maintained the cost of liquid oxygen nearly constant over the past two decades, while foundry raw materials and labor costs have roughly doubled. In light of these recent developments it can be seen that adopting continuous oxygen enrichment techniques, which require a relatively low initial capital outlay, can enable even the small, low output foundry to obtain significant net cost savings.

DESCRIPTION OF CONTROL SYSTEM

Oxygen can be added to the cupola air most conveniently by inserting a small bent pipe into the blast main, downstream of the metering orifice used to measure air flow. This pipe is of a specific size, depending upon the oxygen flow desired, with a closed end and a series of drilled holes beyond the bend. These holes are symmetrically positioned so that the jets of oxygen will enter the blast main at right angles to the direction of the air flow. The drilled section is inserted into the blast main parallel to the air flow and then welded in place. Turbulence in the wind main and at the tuyeres is generally sufficient to provide mixing of the oxygen with the incoming blast air.

Due to the highly oxidizing nature of pure oxygen the control system incorporates an interlocking device which is activated by the flow of air to the cupola. In the event of sudden electrical or mechanical shutdowns or wind spills a solenoid valve is activated to stop the flow of oxygen. Turning on the wind automatically reestablishes the oxygen supply at a pre-set air flow rate.

It should be noted that an oxygen enrichment control system such as the one described above (and illustrated in Figure 15) can, in some cases, be purchased from the manufacturer as a complete

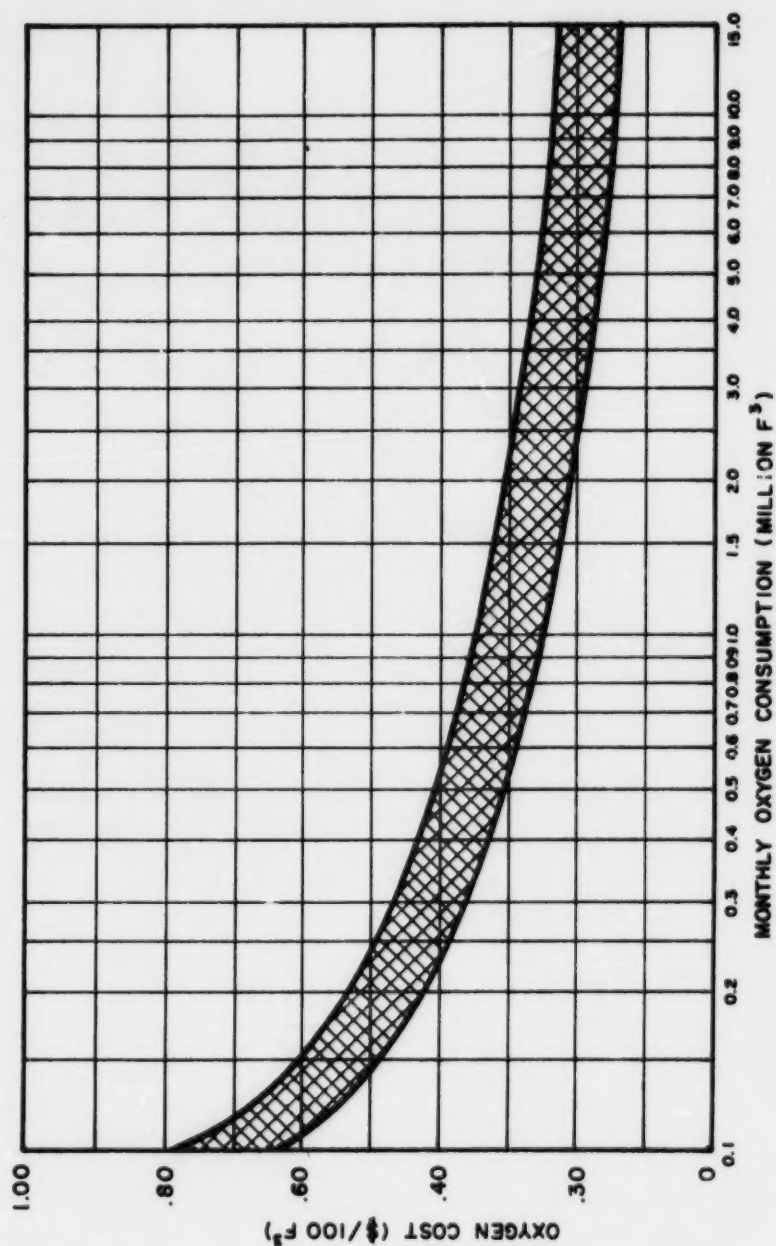


Figure 14. Oxygen cost vs. consumption .

Courtesy of Air Products and Chemicals

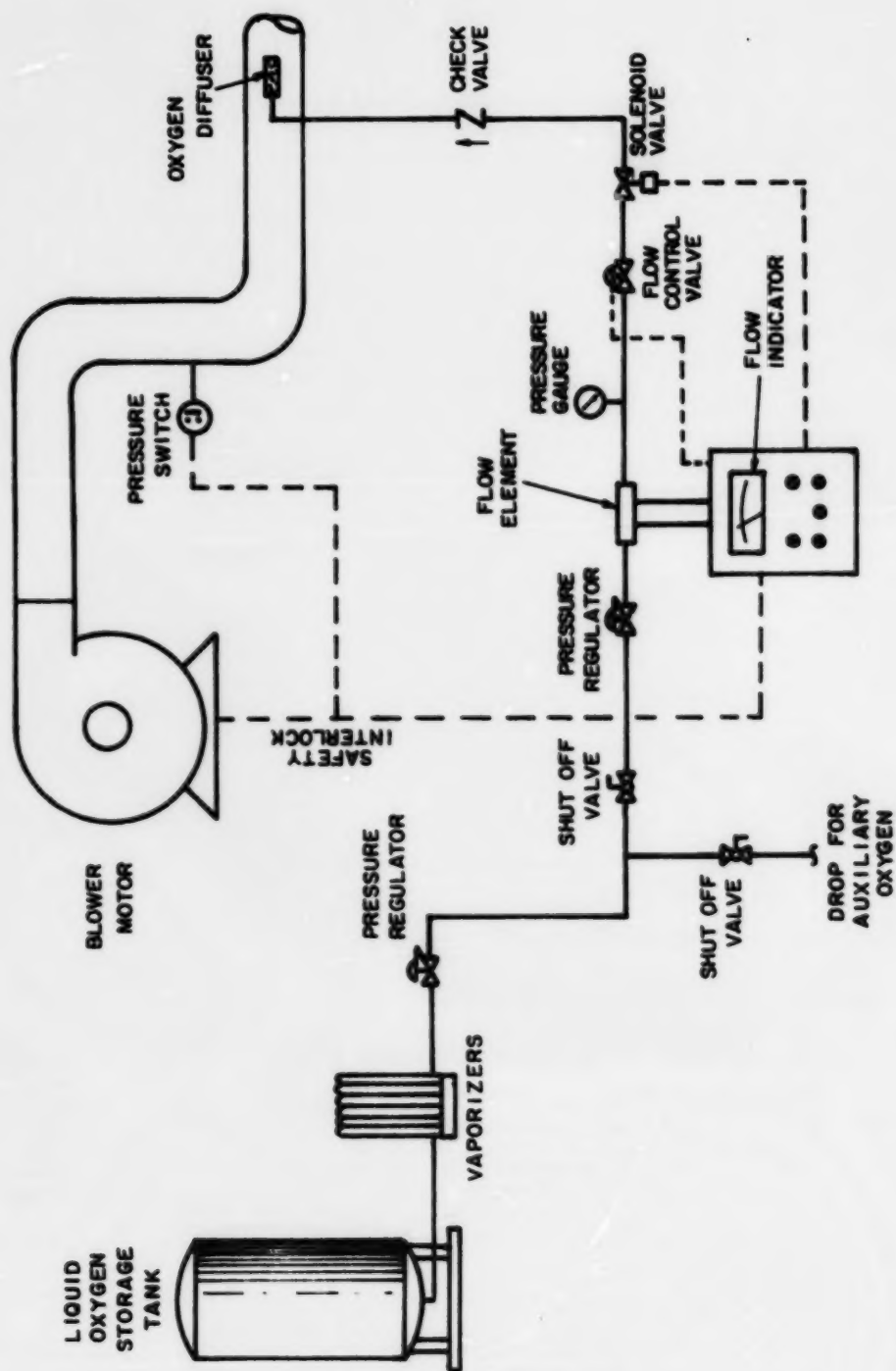


Figure 15. Typical oxygen flow control system.

package. It can be installed without disturbing normal cupola operation and without modifications to the cupola itself. The initial capital investment is nominal when compared with the savings that can be realized by its effective implementation. In addition, there is no extensive training period or additional crew required to operate the flow control system. The procedures involved can be easily and quickly learned and the manufacturers generally offer instructional programs with the sale of their system.

INDUSTRY SURVEY

There are numerous industrial firms involved in the manufacture and supply of liquid gases. Of these companies, however, only a small number are actively engaged in the design and development of the oxygen enrichment systems used in grey iron foundries. In the early stages of this project, communication was established with a number of those companies which are presently marketing systems of this type.

One of the first companies contacted was Airco Industrial Gases, located in Murray Hill, New Jersey. Airco has been promoting oxygen enrichment technology for several years now. They do not manufacture package systems, but they will design the system layout and supply all the necessary hardware to meet plant specifications. They will also provide technical back-up during the installation and start up period and will instruct the operating personnel in correct procedures for operation, maintenance and repair of the associated equipment.

Our initial contact was with a Mr. J.W. Estes who is the manager of combustion metallurgy and development at Airco and has contributed much to the development of the oxygen enrichment technology being utilized in the Airco systems today. Mr. Estes and Mr. J. Blessing co-authored an informative paper on "The Potential for Oxygen Enrichment of the Cupola in the Small Foundry." (8) In this paper are included some pertinent facts of interest to foundrymen who are considering economic and operational alternatives to compensate for the recent exponential rise in coke prices.

The most obvious benefit derived from the implementation of oxygen injection is to effectively increase the average charge ratio (metal/coke). Airco has conducted a number of in-depth studies to compare the before and after operating parameters of cupolas incorporating oxygen enrichment systems developed by Airco. In these studies coke reductions of 40 pounds per ton to as much as 70 pounds per ton of metal charged have been obtained. (8) Oxygen enrichment also reduces the effective stack height of the cupola. Many cupolas have stacks too short for efficient preheating. In striving for maximum production operators often increase pressures at the windbox, forcing a greater volume of combustion gases up the stack. This practice results in excessive heat losses and large top fires. Oxygen injection corrects these conditions by increasing the flame temperature which shortens

the coke combustion zone. This permits a reduction in the total wind volume which effectively maintains the sensible heat in areas where it can perform useful work.

Mr. S. Fredricks, Sales Manager of the Combustion Metallurgy and Development Division at Airco, points out "that other benefits that make oxygen injection profitable are higher steel-to-scrap charge ratios vs. pig iron and castings, and the use of large section sizes. Of course, this is only true if the cupola is operated to make use of the increased carbon pickup and spout temperatures possible with oxygen."

Steel scrap melts at higher temperatures than iron scrap or pig iron. Much more carbon must be dissolved from the coke to finish the iron with a proper analysis when higher steel charges are used. With oxygen enrichment, steel scrap can be increased to as high as 50% of the charge without an abnormal increase in coke. Oxygen injection also allows the use of heavier than normal metal sections. Conventional operation with large scrap, such as ingot molds, ingot butts or large motor blocks, results in chilled metal when the pieces reach the cupola well only partially melted. According to Airco, oxygen enrichment results in complete melting of these sections without chilling the tapped iron. Other difficult-to-melt materials such as briquetted borings can be melted with lower losses.

Another interesting point brought up by Mr. Fredricks is the fact that, in his opinion, "...the biggest incentive to use new methods for reducing foundry costs and to increase production have been pollution control devices."

Common problems encountered in normal cupola operation are bridging of charges, channeling immediately above the tuyeres, and non-uniform combustion. The latter is probably caused by the varying back pressures created by the tail-end systems during the course of a melt. Nothing can substitute for a well calibrated air volume control system, however, enrichment of the cupola air with only 2 to 4 % oxygen can help to eliminate bridging and channeling. By raising the percent of oxygen per cubic foot of air the blast rate may be lowered by as much as 16% without changing the total oxygen content. (8) This wind reduction promotes much more stable combustion and, as pointed out earlier, the lower air volume can be invaluable when the existing emission control system is incapable of handling the full wind rates of the cupola.

Mr. Estes suggests (8) that the most effective way to learn the response of the cupola to oxygen additions is by gradually increasing the level of enrichment. Response of the cupola to oxygen is not linear but occurs in steps. The first increment to 1½% will seem to have little effect, but at 2% the effectiveness will increase rapidly and become most apparent at 2½ to 3%. The cupola

should be operated for a full day at 1½, 2, 2½ and 3% enrichment to learn the most effective level for a given cupola operation. Coke cuts will be necessary for maintaining a uniform temperature and chemistry at the 2, 2½ and 3% levels. Operation beyond the 3% level is not recommended unless the demand for additional melt rate can justify the additional oxygen. In most cases, the most effective coke savings can be obtained at the 2½ to 3% enrichment level. A number of interesting observations have been made concerning the effect of oxygen on coke rate, melt rate, temperature, chemistry and lining erosion.(8)

Coke Rate

A metal/coke ratio is selected to provide the proper spout temperature and the specified carbon pickup for a given charge make-up. Low metal/coke ratios are generally required for high tap temperatures and high steel content charges. At high metal/coke ratios the resultant spout temperatures will be low and carbon pickup considerably less, thus requiring charges containing pig and scrap iron in order to ensure sufficient carbon pickup. Oxygen injection, however, will allow lower coke rates with reductions of 20% being possible using 2½ to 3% enrichment. (8) The reduction in coke rate also has a distinct effect on melt rate since the same coke can melt more charge in a given time.

Melt Rate

Melt rate is determined by metal/coke ratio and rate of combustion is determined by the oxygen driving force. The addition of oxygen to a level of 23.9% O₂ (normal O₂ level in air is 20.9%) in the wind increases the total oxygen content in an amount equivalent to a 19% increase in wind rate. This oxygen enriched blast tends to raise the flame temperature of the combustion zone which in turn accelerates the melting process. A 20% reduction in coke with the same wind rate also increases the melt rate proportionately since 20% more metal can be melted with the same coke.(8)

Temperature

Temperature is also related to the metal/coke ratio and the combustion conditions in the cupola. The use of continuous oxygen will result in elevated temperatures unless adjustment is provided by reduction in the coke and/or blast rate. In foundries where production demands require extended heats and where the tap metal must be consistently maintained within certain design specifications several test runs are generally conducted to arrive at the most effective air/coke rates and tap temperature for that particular operation.

Chemistry

Silicon loss is more pronounced with low metal temperature and high coke beds and thus enrichment can be useful in reducing

silicon loss in acid lined furnaces. The use of continuous enrichment in amounts of 2 to 4% can reduce silicon losses sufficiently to allow a reduction of 50% ferrosilicon, equivalent to 5 lb per ton which could add up to a sizeable savings in raw materials costs over the course of a year's operation.

Scrap Loss Reduction

With the increasingly high costs of capital investments such as pollution control systems, manpower and raw materials, reduced scrap loss is an important consideration to reduce operating costs. With oxygen enrichment even the first metal out of the cupola is often hot enough to form thin-section castings, and need not be pigged for reuse. Airco has found that in foundries producing specialty castings, the saving of one casting per month can be sufficient to pay the entire month's oxygen fee.

Lining Erosion

Since erosion of the cupola lining is a function of temperature and time, it can be stated that a substantial increase in spout temperature can be expected to result in increased lining wear. If the temperatures are kept at the normal level by reducing the blast rate and/or the coke charge the lining wear will be no greater with oxygen, but the profile of wear may change somewhat. Many acid lined cupolas are operated successfully with continuous enrichment for as long as 17 hours on a routine basis; this includes flush tuyere cupolas with limited wind penetrations.

Overtime Costs

Occasionally it has been found that smaller foundries require certain operating personnel to perform two functions. For example, certain individuals may make molds in the morning and pour in the afternoon. In this case, any delays will compel the foundry to hold the crew over for a few hours. The higher metal temperatures and increased melt rate provided by oxygen enriched combustion air might make up for the loss of time.

Included in the Airco study were a series of interesting case histories which illustrate the operational benefits of oxygen enrichment which are being utilized by eight small foundries equipped with Airco injection systems. These results typify the successful application of the oxygen enrichment process in this segment of the grey iron industry.

Typical Case Histories - Airco (8)

Foundry A - Melt rate originally 9 T/hr
42" lined cupola - cold blast
Metal/coke ratio 6.6:1

Foundry A realized a melt rate increase of approximately 20% with an enrichment level of 2% while an increase of 30% was achieved at the 3% enrichment level.

In many smaller foundries, it is not unusual for molders to assume that their work is complete when the pouring operation begins. In some cases, the pouring is done by some of the molders. The substantial increase in melt rate achieved by this foundry with the resulting reduction in O.T. wages paid for the annual oxygen expenditure in one month. In addition to the labor savings, use of FeSi was reduced by 30% while coke reduction amounted to 17.5%. The most significant advantage realized by this foundry was the large reduction in overtime wages.

Foundry B - Melt rate originally 11.5 T/hr
54" Cupola cold blast
Metal/coke ratio 6.6:1

An increase in the melt rate is not necessarily a desired feature in every application of oxygen enrichment. This particular foundry had other objectives which were of greater importance.

Coke was in short supply to such a degree that a 20% reduction in shipments was experienced. At a 2½% enrichment level, this foundry was able to operate a full five days on four days supply of coke.

Since no increase in production was required, this same enrichment level (2½%) allowed a reduction in wind of 10% which in turn lightened the load on the dust collection system.

The most profitable return on the investment in oxygen to this foundry was achieved by the total elimination of stack mounted afterburners. These afterburners were fueled with natural gas which was rising rapidly in cost with the additional danger of curtailed use. They are no longer required. A reduction in the use of ferrosilicon was also achieved.

Foundry C - Melt rate originally 8 T/hr
48" lined cupola - cold blast
Metal/coke ratio 6.6:1

This is another example of a foundry required to operate five days on a four-day supply of coke, which was achieved with a 2½% enrichment level.

Additional cost reductions were realized by substituting cast scrap (purchased and returned) for the more expensive pig iron. No change was observed in metal chemistry.

This particular foundry had a normal scrap loss of 15% of finished castings. Oxygen enrichment has helped to reduce this figure to 6 to 8% by offering increased metal temperature.

Foundry D - Melt rate originally 6.5 T/hr
48" lined cupola - cold blast
Metal/coke ratio 6.6:1

Here is an instance where coke reduction, increased melt rate and the reduction of FeSi use were of minor importance. This foundry specialized in the production of castings weighing approximately 7 tons each. To pour castings of this size required the use of two large ladles permitting simultaneous pouring. The increased metal temperature obtained by an enrichment level of 2% (approximately 1100°F) has reduced scrap loss by 65%.

Foundry E - Melt rate 6 T/hr
36" lined cupola - cold blast
Metal/coke ratio 6.6:1

The single purpose of oxygen enrichment in this foundry was to maximize coke reduction. A 2½% enrichment level permitted a coke savings of 22%.

Foundry F - Melt rate 11 T/hr
54" lined cupola - warm blast
Metal/coke ratio 8:1

The foundry had used oxygen on an intermittent basis for a number of years, primarily for faster recovery following lunch hour, coffee breaks, etc.

While maintaining a metal temperature of 2750°F and a melt rate of 11 T/hr, the following changes were permitted with an enrichment level of 2½ to 2¾%.

1. Wind volume was reduced by 16% resulting in reduced bag house loading.
2. Coke consumption was reduced by 18%.
3. A minor substitution of a portion of the pig iron with costly scrap was also achieved.

Foundry G - Melt rate 5 T/hr
36" lined cupola - cold blast
Metal/coke ratio 7.2:1

The use of oxygen enriched wind probably had the greatest impact on any foundry in the region at this particular location. Airco was able to provide a combined coke reduction of 40% by means of reducing bed height and adjusting the iron-coke ratio. A 50% reduction in scrap loss was achieved by providing continuously hotter metal on a more consistent basis even after temporary wind stoppages.

Proper pollution control had previously required the use of oil fired afterburners. Due to increased combustion efficiency these burners were eliminated resulting in substantial reduction in operating costs in view of the rapidly escalating cost of fuel oil.

Another company which appears to be the leader in the development of oxygen injection technology is Air Products and Chemicals, Inc. in Allentown, Pennsylvania. They are primarily involved in the manufacture and supply of liquified gases for commercial and industrial use, however, their Applied Research & Development Group has been working closely with foundrymen to create cost effective alternatives to compensate for the spiralling rise in raw materials costs and the cost of pollution control equipment. A field trip was arranged to APCI's Corporate Headquarters in Allentown, Penn. Mr. D.R. Ruprecht directed the day's activities which included the inspection of the various in-house design, testing and fabrication facilities utilized in the development of various oxygen enrichment systems, along with a brief inspection of the countless other cryogenic devices being developed for a broad range of industrial applications. Following the tour a conference was held in which a number of pertinent questions were resolved and certain misconceptions regarding oxygen enrichment systems were corrected. The afternoon was spent touring a nearby foundry which is presently utilizing an oxygen injection system which was designed, fabricated and installed by APCI.

Among the major topics of discussion was the question of safety. The reluctance of some foundrymen to adopt oxygen enrichment is understandable when one considers the highly oxidizing nature of pure oxygen. APCI is well aware of the potential hazards involved with the use of oxygen and has gone to great lengths to ensure the integrity of their systems. As an example of the type of precautionary measures considered by APCI Mr. Ruprecht pointed out that in large systems where oxygen is submitted to high velocities special care is taken in selecting the materials to be used in the piping system to avoid the possibility of sparks produced by ferrous particles entrained in the supply stream. In addition, regardless of system size or complexity, all piping and related accessories are chemically cleaned and dried prior to assembly. All valves and fittings are selected for their ability to prevent leakage and minimize wear. Interlocks for protecting against electrical failures and temporary wind spills are standard equipment in all APCI oxygen systems and in the interest of keeping component costs down Mr. Ruprecht has stated that on the larger injection systems the solenoids used are installed on the instrument air lines which provides a safe, positive emergency shut-off while eliminating the need for costly in-line shut-off devices.

Another interesting point that was clarified during our discussion was the fact that the large liquid oxygen storage vessels and the accompanying vaporizing systems do not have to be purchased by the user. This is a point that has often been misunderstood by many foundrymen, because in the course of the literature search no mention was made concerning this. The only cost for which the perspective user is responsible is for the concrete pad on which the storage tank and vaporizing equipment are mounted and a monthly rental fee which covers installation and maintenance both of which are performed by the supplier.

Mr. Ruprecht also pointed out that the average pressure of the storage system is approximately 120 psi which is much lower than the pressures found in oxygen bottles commonly used in the foundry's repair shop which are often well over 2000 psi. Another bonus which can be derived from a conversion to oxygen enrichment is the opportunity to use the oxygen supply for various auxiliary plant applications such as lances, oxyactelene burning and countless other uses and at a cost considerably lower than if purchased in individual cylinders. In addition APCI will conduct a study of the average oxygen consumption rates and will establish a regular supply and maintenance schedule to ensure continuous operation of the enrichment system.

Mr. Ruprecht co-authored an informative paper with Mr. A.L. Love which was published in the AFS Transactions. (6) In this paper he points out that oxygen enrichment can be applied in three basic ways:

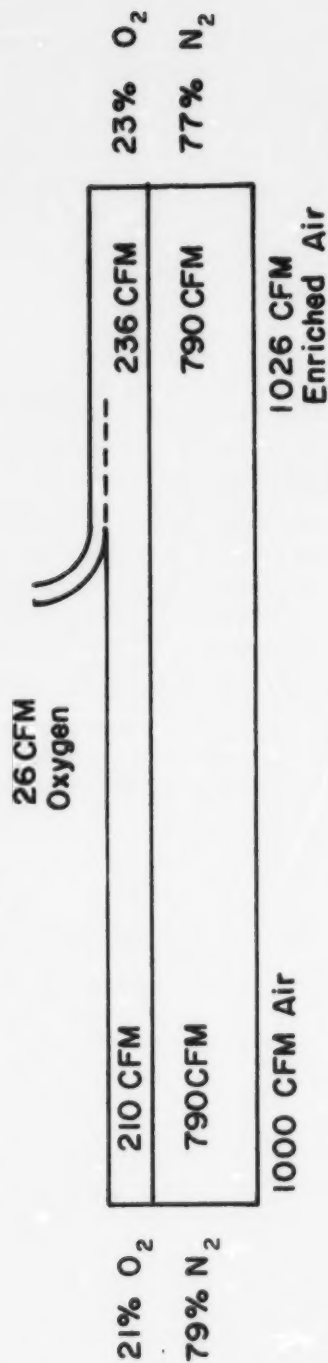
1. Continuously to optimize raw material savings
2. Continuously to increase production
3. Intermittently for quicker startups and greater cupola control.

The foundryman maximizes the benefits with each practice by controlling three variables: wind rate, oxygen content of the wind and the period of enrichment (continuous or intermittent use). These benefits are achieved to greater or lesser degree depending on the method of application and the operating parameters of the cupola.

PRODUCTION INCREASE APPLICATIONS

In general, the minimum production rate required to economically justify a production increase using oxygen enrichment is approximately 15 tons per day, 3 days per week. Oxygen enrichment can provide production increases of 25% and more. (6) This is achieved through enrichment by simply increasing the flow of oxygen to the cupola. More oxygen results in higher combustion rates thus higher melt rate. For example, a cupola operating with a wind rate of 1000 SCFM (standard cubic feet per minute) contains 210 SCFM of oxygen (Figure 16). A 2% enrichment practice would add

Supplemental Enrichment



Equivalent Enrichment

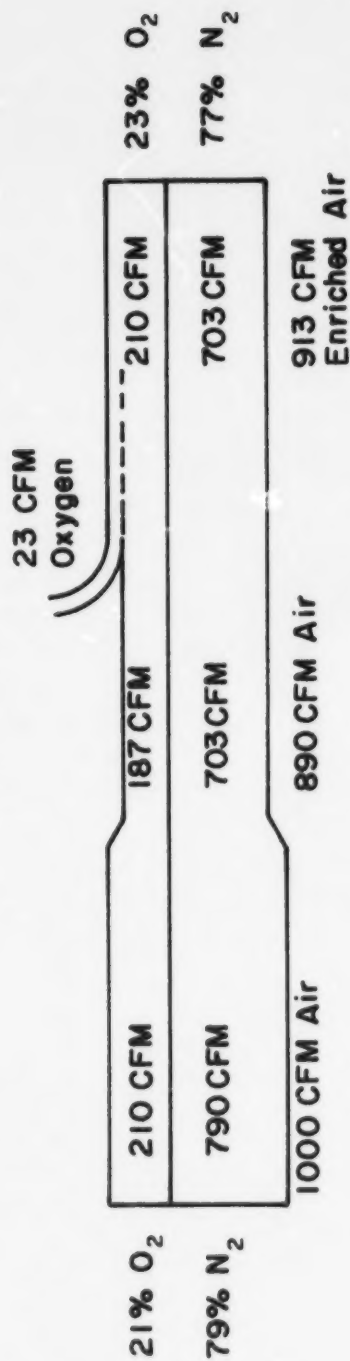


Figure 16. Oxygen enrichment techniques.

Courtesy of Air Products and Chemicals

26 SCFM of oxygen to increase the oxygen content of the air to 23%. A non-enriched wind rate of 1124 SCFM would be required to achieve the same total oxygen flow provided in the 1026 SCFM of enriched air. The difference, of course, is heat absorbing nitrogen. A 2% continuous enrichment practice can provide a 10% to 15% increase in production while a 3% practice offers 20% to 25% increased production. As oxygen enrichment levels are increased beyond 5%, however, production gains become less significant.

Oxygen enrichment may be used intermittently to increase cupola melt rate quickly after shutdowns and during periods of heavy casting line demand. During periods of extremely high casting line demand, the oxygen enrichment level may be raised from 4% to 5% providing an increase in melt rate of as high as 30%. This type of intermittent enrichment practice is used primarily to avoid casting line downtime due to lack of hot metal.

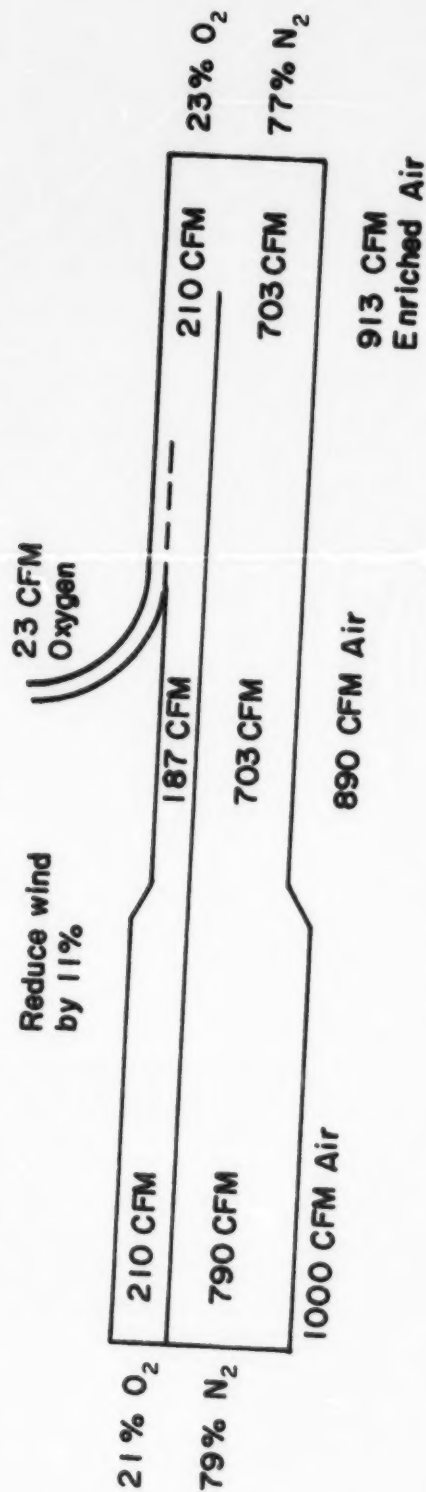
RAW MATERIAL SAVINGS APPLICATIONS

Again, a cupola must produce more than fifteen tons of iron per day and operate at least three days per week to economically justify oxygen enrichment with raw materials savings. When additional melt rate is not required, the oxygen enrichment practice can provide substantial savings in raw material consumption. An equivalent wind enrichment practice would be employed to maximize raw material utilization without increasing the melt rate.

The most common reduced wind enrichment level is 2%. Our sample 1000 SCFM wind rate is again used to illustrate the practice (Figure 17). The wind would be lowered to 890 SCFM and 23 SCFM of enrichment oxygen added. This practice would maintain the same melt rate achieved with 1000 SCFM air, but with 9% less total volume entering the cupola. Eliminated is 87 SCFM of nitrogen, which, as discussed earlier, is a thermal burden on the cupola. By improving the thermal efficiency of the cupola, oxygen enrichment offers several areas for raw material savings: (9)

1. Coke Savings - The increased thermal efficiency of the cupola allows a substantial reduction in the amount of coke needed for smooth operation. A 2% enrichment practice conservatively offers a 10% reduction in coke consumption. This savings alone normally offsets all or a large part of the oxygen cost.

2. Calcium Carbide (CaC_2) Reduction - Calcium carbide is used in the cupola to increase metal temperature and increase melt rate. However, it is very expensive and becoming increasingly harder to obtain. With an oxygen enrichment practice, calcium carbide consumption can be significantly reduced. Savings of 50% or more may be realized due mainly to the increased temperatures that are achievable with oxygen enrichment.



$$\frac{210 \text{ CFM OXYGEN}}{913 \text{ CFM Enriched Air}} = 23\% \text{ or } 2\% \text{ Enrichment}$$

Figure 17. Equivalent enrichment.

3. Pig Iron Substitution - Pig iron is a carbon and silicon rich, but expensive charge material. With oxygen enrichment many foundries have been able to substitute lower-grade, less expensive raw materials for a portion of the pig iron charge. This is made possible through the higher temperatures and improved heat transfer and thermal efficiency achieved in the combustion zone with oxygen enrichment. The substitution of cast scrap for pig offers two significant areas of potential savings. Experience has shown, however, that the substitution of large quantities of steel scrap for pig iron may require the addition of small amounts of silicon bearing charge additions such as FeSi to provide proper metal chemistry. The net result, however, is usually a substantial reduction in total charge.

ADDITIONAL BENEFITS OF OXYGEN ENRICHMENT (9)

Improved Cupola Control

Variations in normal cupola temperature and chemistry can occur frequently due to differences in scrap makeup and size, quality of coke and the moisture content of the air. Adjustments in the oxygen enrichment level can compensate for these variances.

Oxygen enrichment can also be a valuable tool for quickly bringing the cupola to normal temperature after start up or interruptions due to mechanical outages. The time to restore a cold cupola to normal operation has been reduced by as much as 50%, resulting in much less pigging and waste of cold, first tap metal.

Reduction in Casting Rejects

Due to the higher temperatures realized with the oxygen enrichment practice and the ability to increase iron temperature as desired, substantial reductions in rejects due to cold metal may be possible. Oxygen enrichment has proven to be particularly effective during start up by bringing the metal to the desired temperature in a shorter period of time. Experience has shown that as much as 50% of rejects due to cold metal can be eliminated.

Increased Metal Temperature

Increasing the percentage of oxygen entering the cupola results in higher temperatures in the combustion zone. The net effect on metal temperature is an increase of 10°F to 50°F or more depending on the practice. This additional temperature can provide more desirable casting characteristics and result in fewer rejects on the casting line.

Greater Slag Fluidity

The increased melting zone temperature produced by oxygen enrichment results in a more fluid slag. Consumption of slag fluidizers

such as fluorspar can be reduced by 50% or more without sacrificing proper slag properties. Oxygen enrichment reduces slag bridging around the tuyeres and results in a smoother running furnace.

In a subsequent study conducted by APCI seven representative case histories were recorded which demonstrate each of the three methods of application and cover the normal range of operating conditions where raw materials savings, production increases or greater cupola control can be profitably achieved with oxygen enrichment. By comparing his operation with those described in the case histories, the foundryman can estimate the magnitude of savings available to him with the oxygen practice. In some of the cases mentioned, foundry management chose to keep actual dollar savings with oxygen enrichment confidential. Sufficient data has been given, however, to effectively evaluate the oxygen technique. Based on this data, minimum production levels for the profitable application of oxygen enrichment were estimated.

CASE HISTORIES (9)

The first three foundries looked to oxygen enrichment primarily to increase cupola melt rate. In addition, foundries two and three also achieved significant raw material savings. (Foundries four and five employed oxygen primarily to reduce raw material costs.) The last case history is an example of a foundry using oxygen as a control tool to increase metal temperature and melt rate during start up and when needed during the operating day.

Oxygen Enrichment for Increased Melt Rate

Foundry #1 - Cupola Size (ID): 86", Water Wall
Wind Rate: 13,000 CFM, 1000°F
Production: 450 T/D, Malleable Iron

This malleable iron foundry was able to increase cupola melt rate by 20% with 3% continuous supplemental enrichment. This represented an additional 90 tons of metal each day. The daily oxygen cost for this practice is \$1,030 or \$11.43 per extra ton melted.

In this particular case, an increase in metal temperature was also achieved which resulted in a small decrease in fuel consumption for the holding furnace.

Foundry #2 - Cupola Size (ID): 60", Acid Lined
Wind Rate: 5500 CFM, 400°F
Production: 270 T/D, Gray Iron

The second example illustrates an operation which needed additional melt rate during most of the day to meet fluctuating casting line demand. Here a continuous 2% supplemental enrichment practice was employed when additional melt rate was needed.

During periods of reduced casting line demand the wind rate was decreased and the oxygen adjusted to maximize coke savings.

Daily production was increased an average of 8% with increases in melt rate as high as 15% during peak casting line demand. Coke consumption was reduced by an average of 4%. Also, normal additions of proprietary flux were reduced by 75% as a result of higher melting zone temperatures with oxygen. The coke and flux savings offset most of the oxygen cost.

Foundry #3 - Cupola Size (ID): 50", Acid Lined
Wind Rate: 4500 CFM, Cold Blast
Production: 40 T/D, Gray Iron

This gray iron foundry needed a 10% increase in melt rate and a 20°F to 30°F increase in metal temperature. They were able to achieve these goals with 2% enrichment. In addition, increased carbon and silicon levels in the hot metal permitted the operator to substitute steel scrap for a portion of the pig iron charge and reduce coke consumption. The pig iron charge was reduced from 30% to 25% of the metallic charge. A small increase in the silicon briquette charge was required to make up for some of the silicon lost in the steel substitution. Coke consumption was reduced by 5% to 10% depending on coke quality. Evaluation of the process economics showed that the savings in pig iron and coke offset the cost the oxygen and resulted in a net savings of approximately \$90 per day.

These first three foundries are fairly representative of foundries using oxygen enrichment to increase cupola melt rate. In addition to a production increase of 20% or more, gray and ductile cupola operators can usually expect some raw material savings, particularly in coke. Coke savings are not normally achieved when increasing melt rate of a malleable cupola due to the inherently lower carbon levels in a malleable operation.

Oxygen Enrichment for Raw Material Savings

Many foundries which do not need additional production but can still benefit with oxygen enrichment. These foundries would utilize the equivalent or reduced wind enrichment practice discussed earlier. An example of one such operation is the fourth case history.

Foundry #4 - Cupola Size (ID): 54", Acid Lined
Wind Rate: 5500 CFM, 900°F
Production: 200 T/D, Gray Iron

Due to coke quality and supply problems, this foundry had to supplement its coke charge with calcium carbide to achieve the desired metal temperature and melt rate. By decreasing the wind

from 5500 CFM to 4900 CFM and enriching with 2% oxygen, the cupola operator was able to reduce coke consumption from 220 pounds per ton of iron to 190 pounds. In addition, the calcium carbide charge (10 pounds per ton of iron) was eliminated while still maintaining desired melt rate, temperature and chemistry levels. Raw material savings totalled \$660 per day with a daily oxygen cost of \$400 resulting in a net savings of \$260 per day.

Foundry #5 - Cupola Size (ID): 60", Acid Lined
 Wind Rate: 5000 CFM, Cold Blast
 Production: 95 T/D, Gray Iron

This gray iron foundry is using oxygen to reduce coke consumption and improve metal temperature. However in this operation, rather than reduce the normal coke splits, it was decided to reduce the number of coke boosters needed throughout the day by smoothing operating fluctuations with oxygen enrichment. Oxygen is used at an average of 2% enrichment with adjustments above and below this level being made as operating conditions dictate. This practice has resulted in the elimination of an average of 1000 pounds of daily coke booster. In addition, the improved carbon pick up resulting from the higher combustion zone temperatures permits substitution of purchased scrap for a portion of the pig iron charge. As can be seen below, coke and pig iron savings offset the cost of oxygen. The higher metal temperatures have resulted in a reduction in cold metal rejects.

	<u>CONVENTIONAL</u>	<u>OXYGEN ENRICHMENT</u>
Coke (lbs/ton Fe)	287	287
Coke Boosters (lbs/ day)	1800	800
Pig Iron (lbs/ton iron)	80	50
Tap Temperature	2780°F	2850°F
	<u>\$/DAY</u>	
Coke and Pig Iron Savings	160	
Oxygen Costs	<u>145</u>	
NET SAVINGS	15	

While most foundries using oxygen enrichment achieve sufficient savings to offset the cost of an oxygen practice, oxygen is not economical for all. Generally, a foundry must be producing more than 30 tons per day to justify oxygen enrichment on raw material savings alone. The following outlines an evaluation made for a small gray iron foundry, which shows that oxygen enrichment would not be economical for this operation.

Foundry #6 - Cupola Size (ID): 42", Acid Lined
 Wind Rate: 2700 CFM, Cold Blast
 Production: 32 T/D, Gray Iron

This foundry did not need any additional melt capacity or metal temperature. The cupola charge consisted of 70% cast scrap and 30% returns with a metal to coke ratio of 10 to 1. In this operation the 10% to 15% coke savings possible with 2% enrichment would offset only 30% to 50% of the oxygen costs as shown below.

	<u>\$/DAY</u>
15% Coke Savings	53
Oxygen Cost	<u>100</u>
NET COST	<u>47</u>

Oxygen Enrichment for Cupola Control

While some operations, particularly smaller foundries, cannot justify the use of continuous enrichment for production or raw material savings, they can usually realize economic benefit using oxygen intermittently for faster start ups, improved operation control and higher temperatures.

Foundry #7 - Cupola Size (ID): 46", Acid Lined
 Wind Rate: 3800 CFM, Cold Blast
 Production: 150 T/D, Gray Iron

The final example outlines the results one such foundry achieved using oxygen enrichment intermittently. Fluctuations in casting line demand at this particular operation made it necessary to go off wind frequently during the day. This resulted in low casting temperatures particularly after these off wind periods. With oxygen enrichment, when the cupola is brought back on wind, 2% to 3% enrichment is used to quickly bring the metal temperatures up to the desired level resulting in a substantial reduction in cold metal rejects. The enrichment practice has also reduced the amount of pigging required during start up by over 50%.

Finally, the higher metal temperatures have allowed this foundry to eliminate their 1000 pounds per day CaC₂ additions representing a savings of \$50 per day. On the average, oxygen is used four hours out of the sixteen hour operating day. Thus, including CaC₂ savings, this foundry greatly reduced pigging and cold metal problems at a net oxygen cost of only \$35 per day. (See Table 13 for a summary of the case histories.)

COST ANALYSIS OF OXYGEN ENRICHMENT SYSTEMS

Figure 18 represents a conceptual view of an oxygen enrichment system. This system can be broken down into three areas:

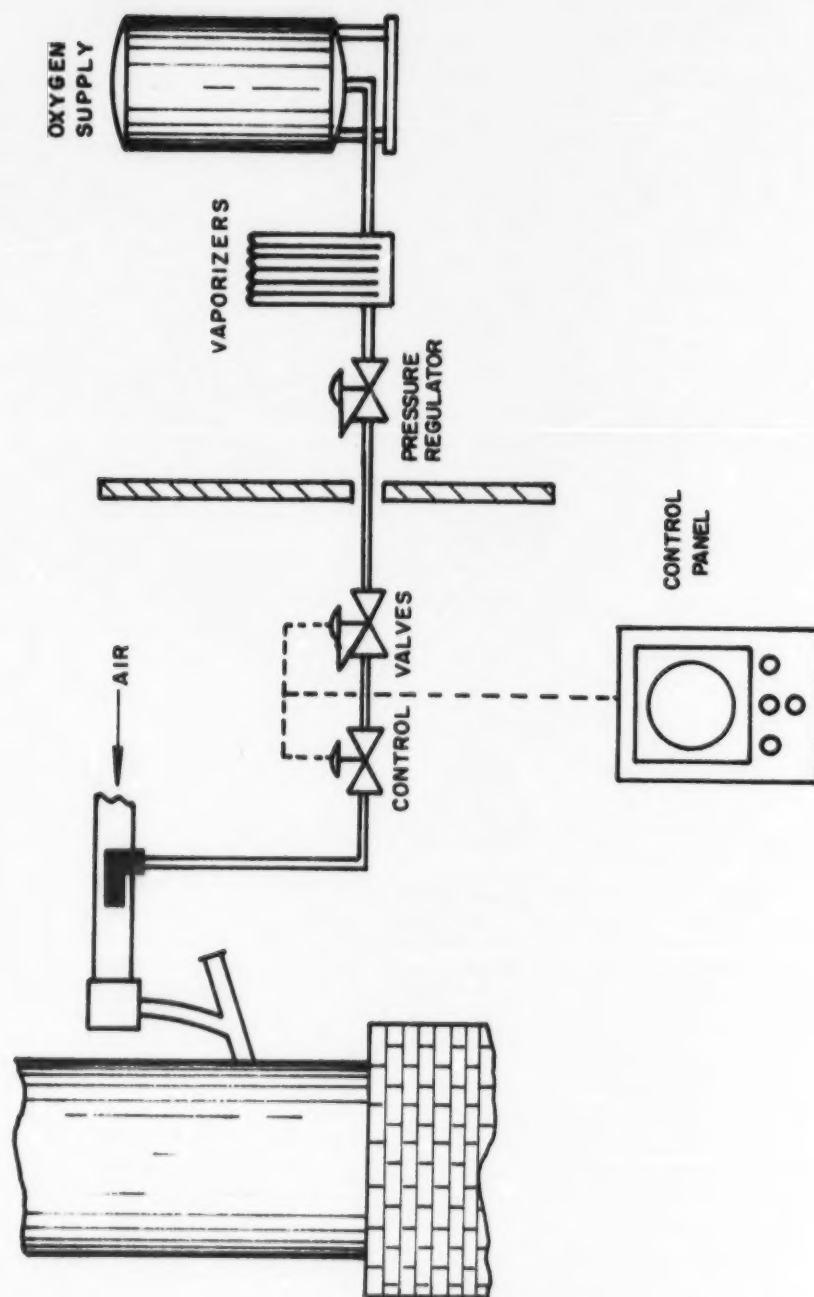


Figure 18. Oxygen system.

Table 13. CASE HISTORY SUMMARY - APCI

	<u>Cupola Inside Diameter</u>	<u>Wind Rate</u>	<u>Production</u>	<u>Savings Area</u>
A. Oxygen Enrichment for Increased Melt Rate				
Foundry #1	86 inch water well	13,000 CFM hot blast	450 T/D malleable iron	20% increase in melt rate
Foundry #2	60 inch acid lined	5,500 CFM hot blast	270 T/D gray iron	8% production increase 15% increase in melt rate 75% reduction in flux additions
Foundry #3	50 inch acid lined	4,500 CFM cold blast	40 T/D gray iron	10% increase in melt rate 5% pig iron reduction 10% coke reduction
B. Oxygen Enrichment for Raw Material Savings				
Foundry #4	54 inch acid lined	5,500 CFM hot blast	200 T/D gray iron	14% coke reduction Elimination of calcium carbide additions.
Foundry #5	60 inch acid lined	5,000 CFM cold blast	95 T/D gray iron	55% reduction of coke boosters
Foundry #6	42 inch acid lined	2,700 CFM cold blast	32 T/D gray iron	15% coke reduction Oxygen enrichment uneconomical.

Table 13. CASE HISTORY SUMMARY - APCI (Continued)

	<u>Cupola Inside Diameter</u>	<u>Wind Rate</u>	<u>Production</u>	<u>Savings Area</u>
C. <u>Oxygen Enrichment For Cupola Control</u>				
Foundry #7	46 inch acid lined	3,800 CFM cold blast	150 T/D gray iron	Reduction of cold metal rejects. 50% reduced pigging Elimination of calcium carbide additions.

1. Liquid oxygen storage and vaporization system
2. Oxygen supply line
3. Oxygen enrichment flow control system

Liquid Oxygen Storage and Vaporization System

Since relatively large volumes of oxygen are needed for oxygen enrichment, a bulk oxygen storage system is normally required. The system consists of a tank, vaporizers and normally some type of pressure regulating equipment (see Figure 19). The oxygen is stored as a liquid (at -300°F) in a double-shell cryogenic tank. In liquid form, a large amount of oxygen can be stored in a relatively small volume (one gallon liquid oxygen equals 115 standard cubic feet of gaseous oxygen).(9) When oxygen is needed for cupola enrichment or oxygen lances, etc., the cold liquid is automatically transferred from the oxygen tank to a vaporizer. The ambient air vaporizer is the most common type. In this unit the liquid oxygen absorbs heat from the surrounding air and changes to a gas. As it changes, it expands providing the pressure needed to deliver oxygen to the cupola area. Pressure is normally regulated before it leaves the tank area.

The only capital investment the foundry makes in this area of the system is for a concrete foundation. The oxygen tank and vaporizer are normally leased from the oxygen supplier. This frees the foundryman from the responsibility of servicing and maintaining the fairly sophisticated cryogenic equipment. In addition, with ambient air vaporizers no power is required except to pump the liquid oxygen from the tank truck to the storage tank during deliveries. When extremely high oxygen flow rates are needed electric or steam assisted vaporizers may be required.

Oxygen Supply Line

From the tank area the gaseous oxygen is piped to the cupola, maintenance shop, or any other oxygen use point. The oxygen supplier will usually provide assistance in the design and sizing of the supply line.

Oxygen Enrichment Flow Control System

The equipment used to control the flow of oxygen into the cupola (and thus the oxygen enrichment level) can range from simple to quite sophisticated. A basic flow control system (see Figure 15) includes:

1. Pressure regulator
2. Flow measuring device
3. Flow control valve
4. Electrical shutoff valve
5. Check valve
6. Diffuser

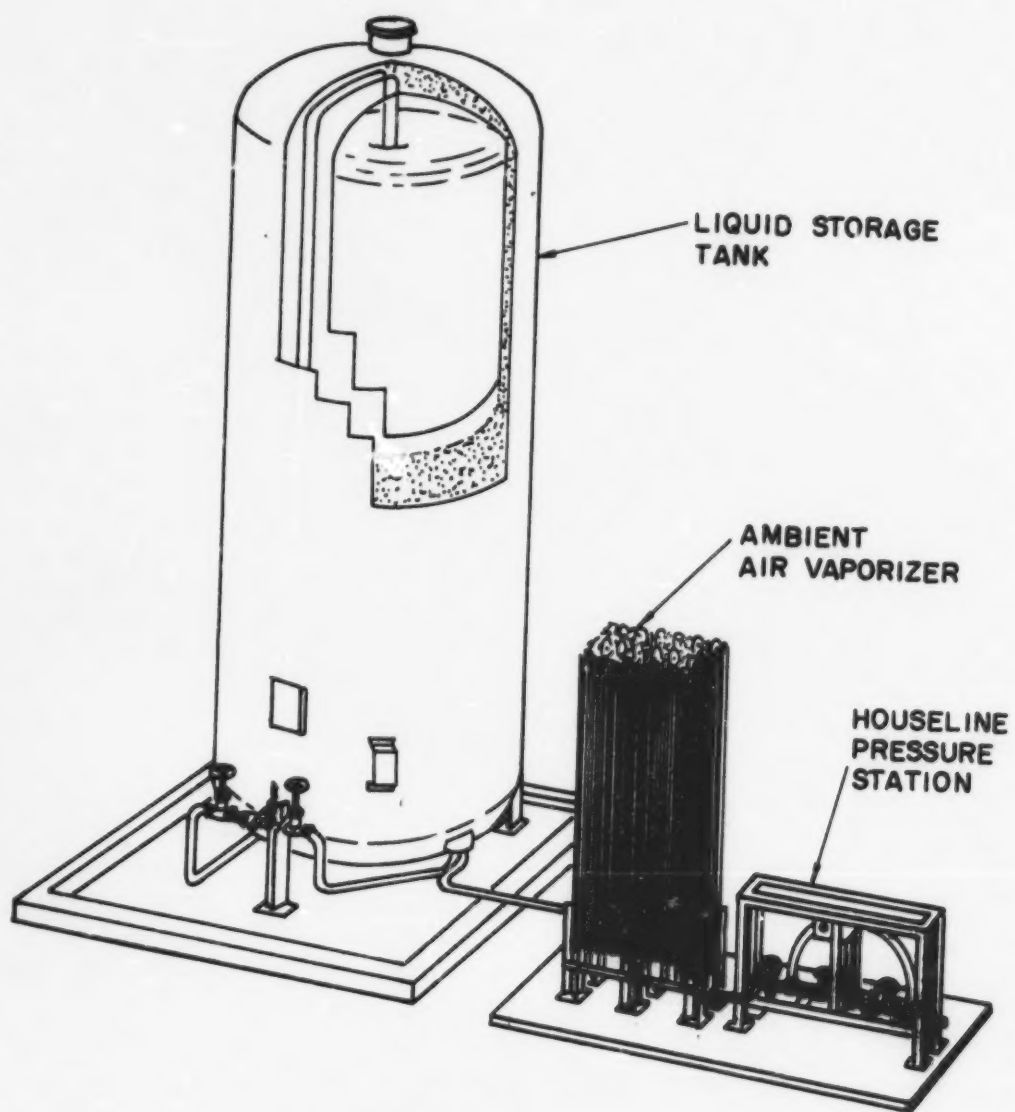


Figure 19. Typical liquid tank installation.

The system is normally interlocked with the cupola air blast to insure that oxygen cannot flow without sufficient air flow.

Foundries operating more than a few hours a day would find a recording type system valuable for establishing and maintaining an economical enrichment practice. If frequent changes in the cupola wind are made during the operating day, an automatic control system may be desirable. An approximate cost range for each type of system is provided below. (9)

	<u>PRICE RANGE</u>
1. Manual System with Flow Indicator	\$1000-\$3000
2. Manual System with Recorder and Remotely Operated Valves	\$3000-\$6000
3. Automatic Ratio Type System	\$5000-\$8000

The wide cost range is due primarily to the size of the flow components required for the maximum oxygen flow rate.

In summary, the foundry's capital investment in the oxygen enrichment process includes:

1. Oxygen Tank Foundation
2. Oxygen Supply Line
3. Flow Control Equipment

For most operations, the total investment is \$5000 to \$20,000. (9)

In order to determine the cost effectiveness of oxygen enrichment, the foundryman must first decide which method of application will best serve his needs. If his cupola produces over 15 tons of hot metal per day, raw materials savings and reduced wind rate benefits are possible with oxygen enrichment. The production increase application is called for if he has unused casting capacity and the extra product can be sold. Regardless of his production level, he can achieve greater cupola control with the oxygen practice.

Next the foundryman must carefully evaluate all the benefits of oxygen enrichment. It is best to calculate the maximum and minimum savings that can be achieved with each benefit. Even intangibles, like the cost of downtime on the casting line, due to the lack of hot metal, should be considered. The case histories previously cited, however, provide a representative picture of savings that might be achieved for similar applications and operating conditions.

Finally, oxygen consumption should be calculated. A rough estimate may be obtained by multiplying wind rate by 2% by the on wind time. For example, the monthly oxygen consumption for a cupola operating at an 8000 CFM (225 m³/min) wind rate and 2% continuous oxygen enrichment with 10 hours per day on wind time would be approximately (8000 CFM (225 m³/min) x 2% x 60 min/hr x 10 hr/day x 20 days/month) = 1,920,000 ft³ (55,000 m³).

Oxygen cost may be estimated using Figure 14. The unit cost of oxygen is primarily dependent on consumption level, but the oxygen price also varies considerably with the distance from source to consumer, the type of vaporization required and other factors. Additionally, U.S., Canadian and European oxygen prices differ. At the 1.9 million ft³ (55,000 m³) per month consumption level of the example, oxygen cost is roughly \$0.22 per hundred cubic feet as derived from Figure 14. Monthly oxygen cost would, therefore, be approximately \$4400.

Comparing total savings to total oxygen cost determines the economic feasibility of oxygen enrichment. Economic evaluations of oxygen enrichment are, of course, greatly dependent on the price of the raw materials saved and/or the price attached to increases in production. Since these may change significantly over a one year period, annual re-evaluation of oxygen enrichment would be prudent. Clearly, the tendency in recent years has been towards the use of oxygen enrichment.

SECTION VII

DIVIDED BLAST TECHNIQUES

INTRODUCTION

The concept of introducing blast air into the furnace through single, double, or multiple rows of tuyeres has been with us for several years. Claims have been made at various times that by the use of two or more rows of tuyeres, higher metal tapping temperatures can be obtained for a given coke charge or, conversely that charge coke consumption can be reduced for a given metal tapping temperature.

An interesting prototype was the balance blast cupola, introduced by BCIRA over forty years ago.(10) The system was constructed so that the relative amounts of air admitted to each of the several levels could be adjusted to give a maximum of carbon monoxide at the bottom and a minimum at the top of the combustion zone, thus utilizing all of the potential heat of the coke.

In principle, the concept of balanced blast appears valid, however, the control and monitoring systems available at that time were incapable of effectively proportioning and balancing the amount of blast among the various rows of tuyeres. It was therefore left to the individual judgement of the cupola operators to establish and maintain optimum conditions throughout the burn.

Though the system realized a limited degree of success it was never adopted on a large scale and most foundries chose to remain with the more conventional single row of tuyeres.

In light of the recent exponential rise in raw material costs, along with the prospects for increasingly stringent pollution control standards, foundrymen have begun to reexamine the possibility of improving cupola operating efficiency and economy by the use of two rows of tuyeres.

A comprehensive series of tests were conducted by the British Cast Iron Research Association (BCIRA) at their cupola plant in Alvechurch, Birmingham, England, in an effort to define the conditions under which these improvements could be obtained. The authors feel that these are probably the most detailed and informative tests available to date and that they are well worth noting in this publication.(10)

TEST PROCEDURES

The cupola used for these tests had an internal diameter of 30 in. (76.2 cm). Two windbelts were provided: one for the supply of air to four lower tuyeres, (each of 4 in. (10 cm) diam.) and the other supplying air to four upper tuyeres of the same diameter.

The distance between lower and upper tuyeres could be varied by the

provision of several alternative rows spaced at one foot intervals, the rows not in use being sealed off. The flow of blast air to each lower and upper row of tuyeres was measured, recorded and automatically controlled.

The cupola was continuously tapped and slagged. The temperature of the metal was measured continuously by a Platinum/Platinum-13% Rhodium thermocouple immersed in the clean metal compartment of the front siphon box and recorded on a strip chart recorder.

The tests were conducted as follows:

Varying the Distribution of the Blast Air Between Lower and Upper Tuyeres

These tests were carried out under three different combined conditions of total blast rate, coke charge and spacing of the lower and upper rows of tuyeres. At each set of conditions, the blast distribution to upper and lower tuyeres was varied. Figure 20 shows that for each set of conditions investigated, increasing the proportion of total blast supply to the upper rows of tuyeres and correspondingly reducing that to the lower row:

1. Increased tapping temperature of the metal until the blast was divided approximately equally between lower and upper rows. On further increasing the proportion of air supplied to the upper tuyeres, temperature decreased.
2. Reduced melting rate slightly.
3. Increased combustion efficiency and stack gas temperature.

Figure 21 shows that increasing the proportion of air supplied to the upper tuyeres:

1. Increased carbon content of the metal, a maximum value being obtained when the blast was equally divided between the two rows of tuyeres.
2. Had little significant effect on manganese and sulfur contents of the metal.
3. Tended to reduce silicon content of the metal slightly.

Varying the Distance Between the Lower and Upper Tuyeres

The results already obtained indicated that for a given spacing of the lower and upper tuyeres, maximum temperature was obtained when the air supply was divided equally between the two rows. A further series of tests was carried out to examine the effects of varying distance between the two rows of tuyeres to determine optimum spacing and to find if this was influenced by the amount of charge coke used and the blast rate.

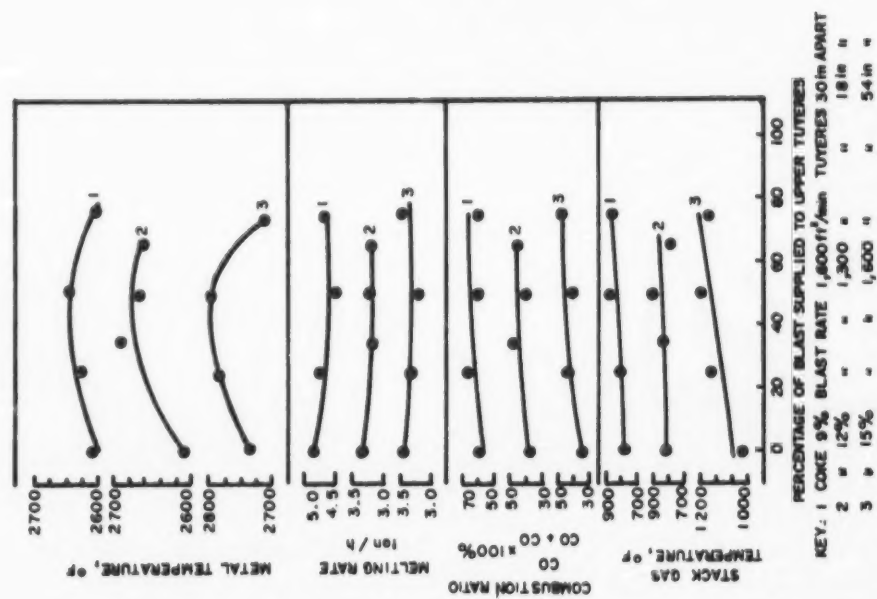


Figure 20. Effect of varying blast distribution to lower and upper tuyeres on furnace performance. (10)

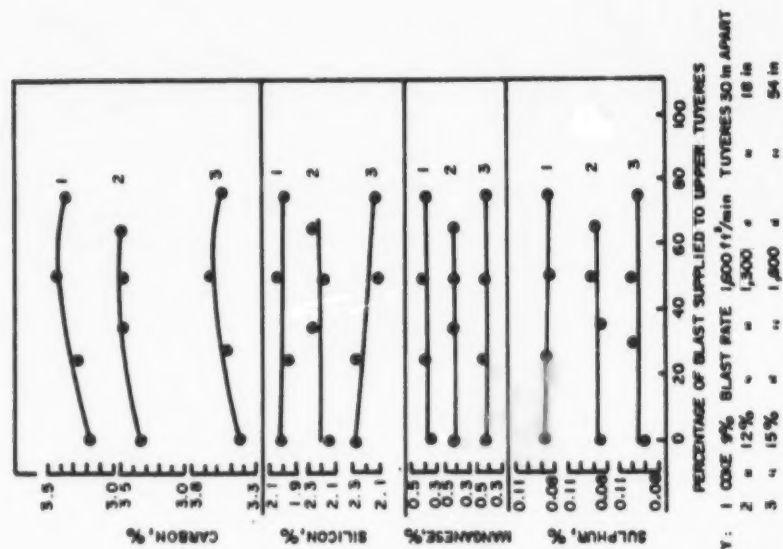


Figure 21. Effect of varying blast distribution to lower and upper tuyeres. (10)

Vary Charge Coke Quantity and Distance Between Upper and Lower Tuyeres at Constant Blast Rate

Tests were conducted at charge coke quantities of 9, 12 and 15% by weight of the metal charge; at each charge coke quantity, the distance between the two rows of tuyeres was varied. In addition, one test was made at a charge coke quantity of 7% with a tuyere spacing of 30 in. (76.2 cm). The blast rate in all tests was 1600 ft³/min (45 m³/min) and this was divided equally between the two rows of tuyeres (except, of course, when only one row was used where upon the total blast quantity was delivered to this row). Figure 22 shows that at each charge coke quantity, increasing the distance between two rows of tuyeres:

1. Increased metal temperature until a maximum value was attained when the tuyeres were spaced between 30 in. (76.2 cm) and 42 in. (106.7 cm) apart (i.e., approximately 36 in. (91.4 cm); when tuyere spacing was increased further, metal temperature decreased from the maximum value.
2. Slightly reduced melting rate.
3. Increased combustion efficiency and stack gas temperature.

Figure 23 shows that by varying the distance between the two rows of tuyeres:

1. Carbon and manganese contents of the metal attained maximum values and sulfur content a minimum value when the tuyeres were spaced approximately 36 in. (91.4 cm) apart.
2. Silicon content of the metal decreased progressively as the distance between the two rows of tuyeres was increased.

Vary Blast Rate with Constant Coke Charge

Results of these tests showed that the optimum spacing of two rows of tuyeres was independent of the amount of charge coke used. To determine if it was influenced by blast rate, tests were carried out at blast rates of 1300, 1600 and 1900 ft³/min. (37, 48 and 54 m³/min.) at various tuyere spacings, the charge coke quantity being kept constant at 12% by weight of the metal. Figure 24 shows that optimum distance between the two rows of tuyeres (i.e., the distance at which maximum metal temperature was obtained) was not influenced by blast rate.

BENEFITS DERIVED FROM DIVIDED BLAST

Figure 25 shows the relationship between charge coke quantity and metal temperature at a blast rate of 1600 ft³/min. (45 m³/min.) for operation with one and two rows of tuyeres. In the latter case the cupola was arranged with optimum spacing of the two rows of tuyeres and with the blast divided equally between them.

This illustration shows that at a given charge coke quantity, the metal temperature increased by 80° to 90°F (45° to 50°C) by operating

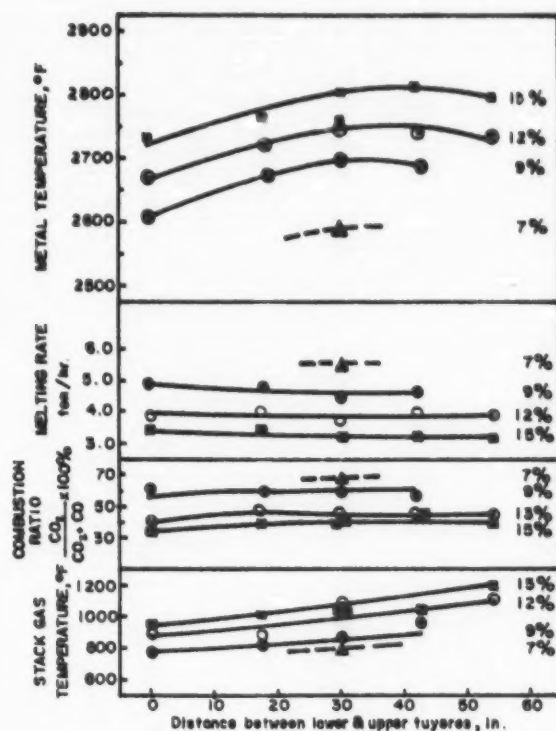


Figure 22. Effect of tuyere spacing on furnace performance at various charge coke quantities (blast rate 1600 ft³/min. = 45 m³/min.) (10)

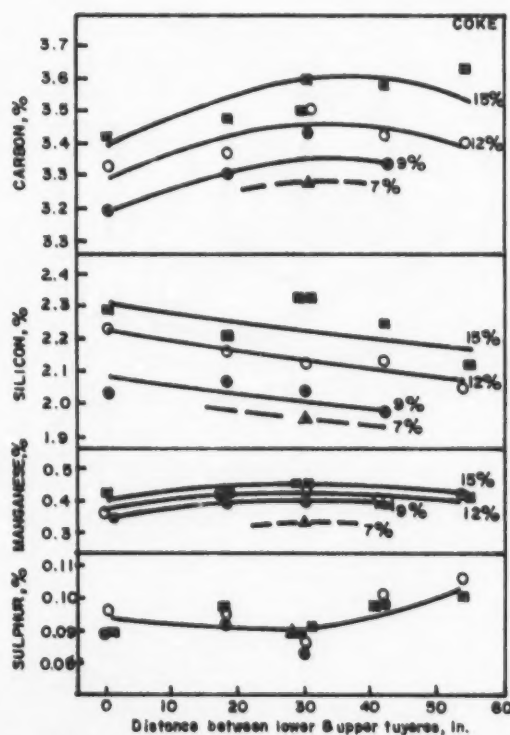


Figure 23. Effect of tuyere spacing on metal composition at various coke charge quantities (blast rate 1600 ft³/min. = 45 m³/min.) (10)

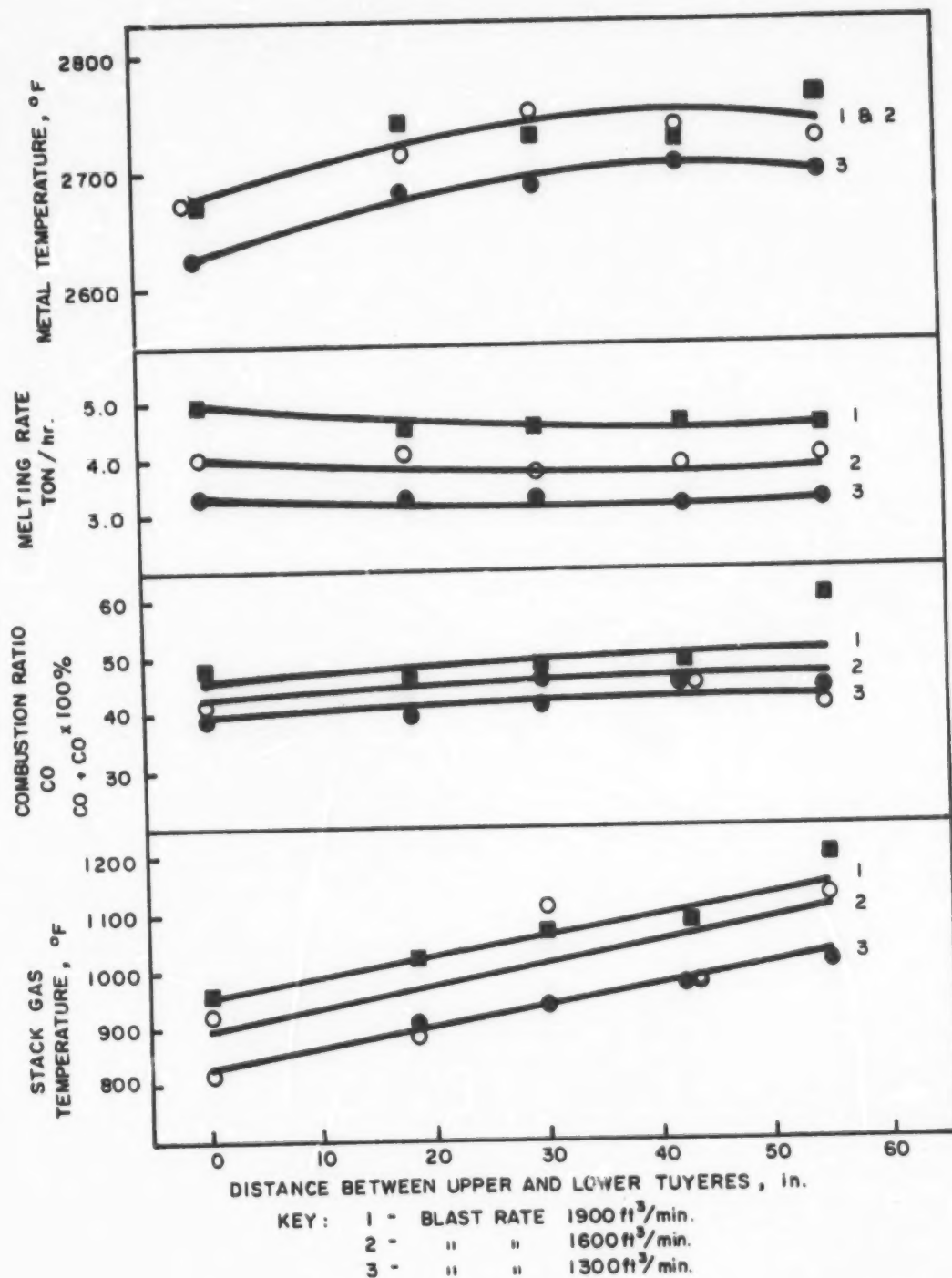


Figure 24. Effect of tuyere spacing on furnace performance at various blast rates (coke charge 12 %).(10)

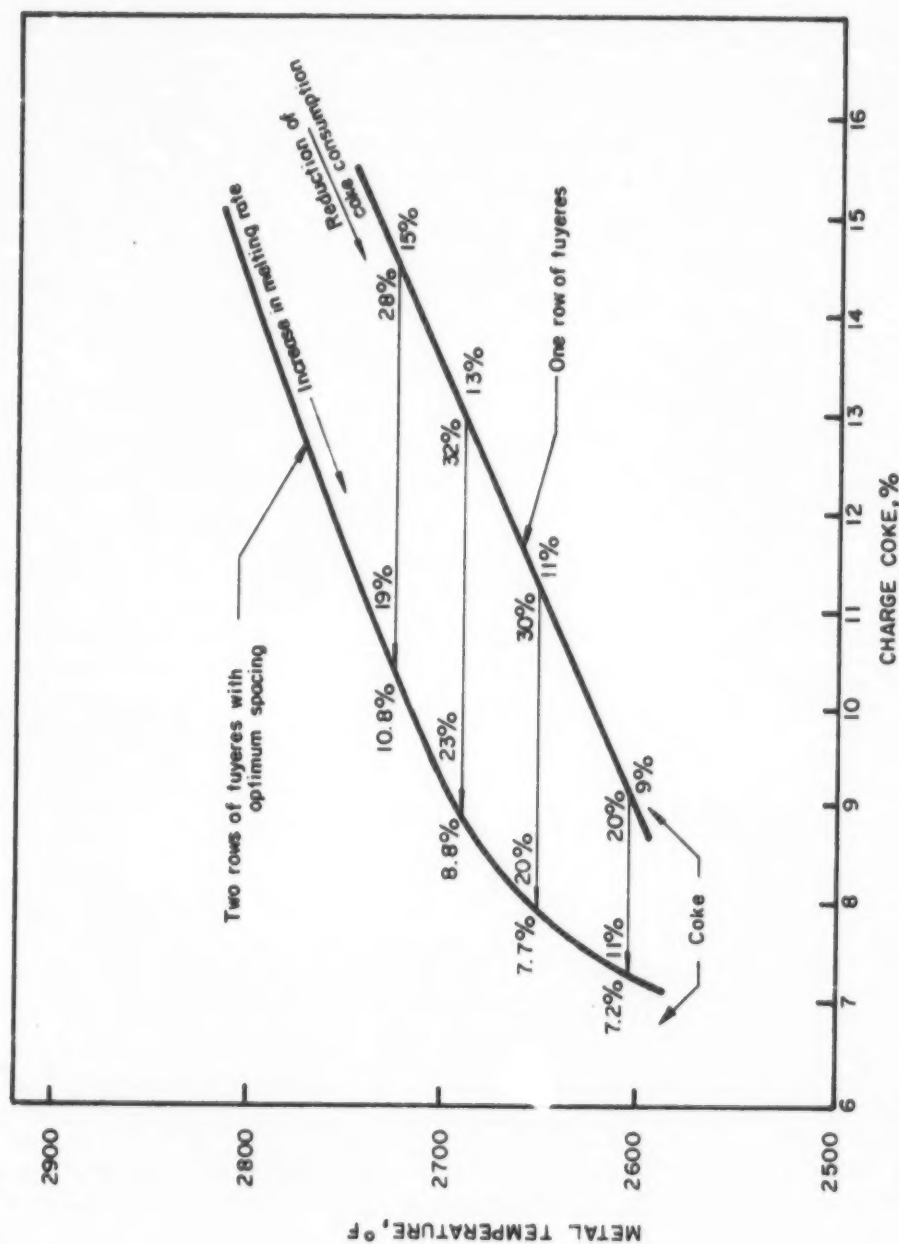


Figure 25. Reduction of charge coke consumption and increase in melting rate by operating cupola with two rows of tuyeres with divided blast supply (blast rate 1600 ft³/min. = 45m³/min.)(10)

with two rows of tuyeres. It also indicates the extent to which charge coke consumption could be reduced for a given metal tapping temperature by using two rows of tuyeres and a divided blast supply. For example, a charge coke consumption of 15% was required to produce a tapping temperature of 2732°F (1500°C) when operating with a single row of tuyeres whereas, with two rows, this temperature was obtained for a charge coke consumption of 10.8%. At a blast rate of 1600 ft³/min (45 m³/min.) (the melting rate obtained with 15% charge coke was 3.41 tons per hour with one row of tuyeres). When using two rows of tuyeres and 10.8% charge coke, the melting rate was 4.07 tons per hour. Thus, operation with two rows of tuyeres allowed charge coke consumption to be reduced by 28% and the melting rate to be increased by 19% compared with operating with one row of tuyeres while permitting a tapping temperature of 2732°F (1500°C) to be obtained. Figure 25 also shows the extent to which charge coke consumption could be reduced and the melting rate increased for other required levels of metal temperature. Within the range studied, the reduction of coke consumption amounted to between 20 and 30% of that required when operating with one row of tuyeres. The corresponding increase in the melting rate was between 11 and 23%.

Effect on Carbon and Silicon Content of Metal

Figure 26 shows the relation between metal temperature and carbon content of the metal at various tuyere spacings, the results relating to the tests carried out at a constant blast rate of 1600 ft³/min. (45 m³/min.) and various charge coke quantities. At each tuyere spacing, carbon content increased with metal temperature but (as shown by the bottom diagram, in which lines on the upper diagram have been combined) for a given metal temperature, carbon content of the metal increased slightly as the distance between the two rows of tuyeres was increased.

Figure 27 shows the relation between metal temperature and silicon content of the metal. As in the case of carbon content, the silicon content increased with metal temperature at a given metal temperature as the distance between the two rows of tuyeres increased.

Thus, by operating with two rows of tuyeres with optimum spacing, carbon content was approximately 0.06% higher and silicon content approximately 0.17% lower than obtained when operating with a single row of tuyeres, metal temperature being identical in both cases. The carbon equivalent value ($C\% + \frac{Si\% + P\%}{3}$) was not therefore

affected by varying tuyere spacing at constant metal temperature.

Optimum Blast Rate

Figure 28 illustrates the effect on metal temperature of varying the blast rate at a constant charge coke quantity of 12% when operat-

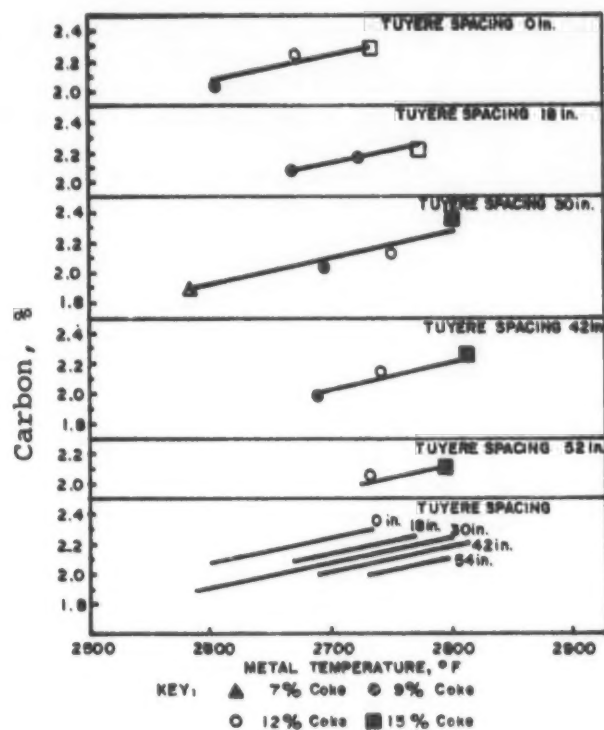


Figure 26. Effect of metal temperature and tuyere spacing on carbon content of metal. (10)

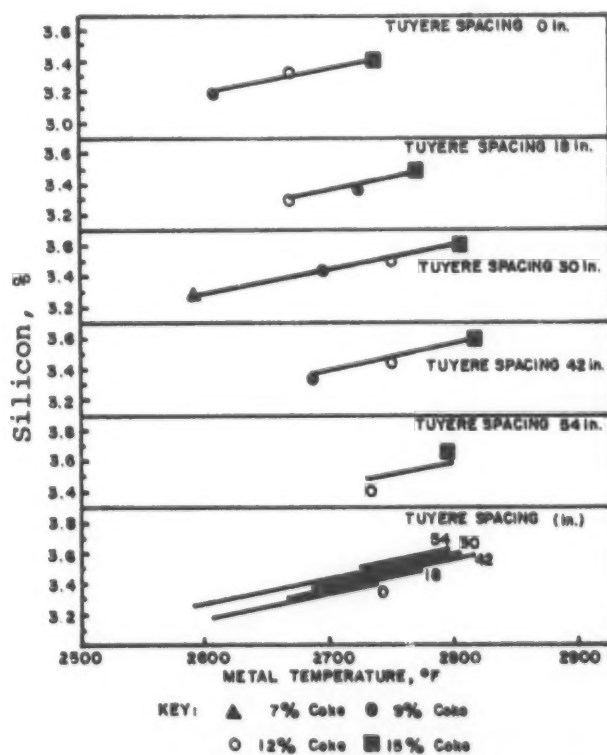


Figure 27. Effect of metal temperature and tuyere spacing on silicon content of metal. (10)

ing with a single row of tuyeres and with two rows spaced 18, 30, 42 and 54 in. apart (47.5, 76.2, 106.7 and 137.2 cm). The curves have been drawn so they are consistent with the relationships between metal temperature and tuyere spacing of various blast rates shown in Figure 24. They show that operation with two rows of tuyeres did not affect optimum blast rate which, at 12% charge coke, was 1750 ft³/min (49.5 m³/min.) corresponding to a specific blast rate of 360 ft³/min (107 m³/min.).

Burn-out Pattern and Spacing of Tuyeres

The burn-out pattern in the cupola studies is illustrated in Figure 29. The burn-out extended further upwards when using two rows of tuyeres but it was not as deep, so that operating with two rows of tuyeres had little effect on the quantity of material required for repair of the lining.

The distance from the center of the tuyeres to the top of the burn-out when operating with a single row of tuyeres was approximately 36 in. (91.4 cm) and varied little with the charge coke quantity or blast rate. It was also observed, that, when operating with two rows of tuyeres, the burn-out extended approximately 36 in. (91.4 cm) above the center of the upper row of tuyeres. This was a very clear indication that the use of an upper row of tuyeres extended the height of the operating coke bed. The improved performance obtained with divided blast is believed to be due to this extension of the coke bed, which improves heat transfer to the molten metal passing through it, rather than to any substantial combustion of carbon monoxide by the air entering the upper row of tuyeres.

The distance between the tuyeres and the top of the burn-out 36 in. (91.4 cm) corresponded very closely to the optimum spacing of the tuyeres. It was therefore thought that this might provide a criterion for spacing the tuyeres on cupolas of varying sizes and practices.

To obtain information on the burn-out pattern in cupolas of different sizes, several foundries were requested to provide details of cupola lining wear. The cupolas varied from 28 in. (70.12 cm) to 87 in. (221.0 cm) internal diameter and covered a wide variety of practice, the charges varying from high pig iron and cast iron scrap mixes to high steel scrap mixes. The height of the burn-out for these cupolas is shown in Figure 30, from which may be seen that this is not a function of the cupola diameter. Additionally, in the great majority of cases, the height of the burn-out was within the range 33 to 48 in. (83.3 to 121.9 cm). Inspection of Figure 22 shows that within this range of tuyere spacings, variation of the spacing had very little effect on furnace performance. It was therefore tentatively suggested that in designing or converting cupolas for operation with two rows of tuyeres, the rows should be spaced between about 30 in. (76.2 cm) and 42 in. (106.7 cm) apart. From experience of cupolas up to 73 in. (185.4 cm) internal diameter converted to operate with two rows of tuyeres, this recommendation appears to have been correct.

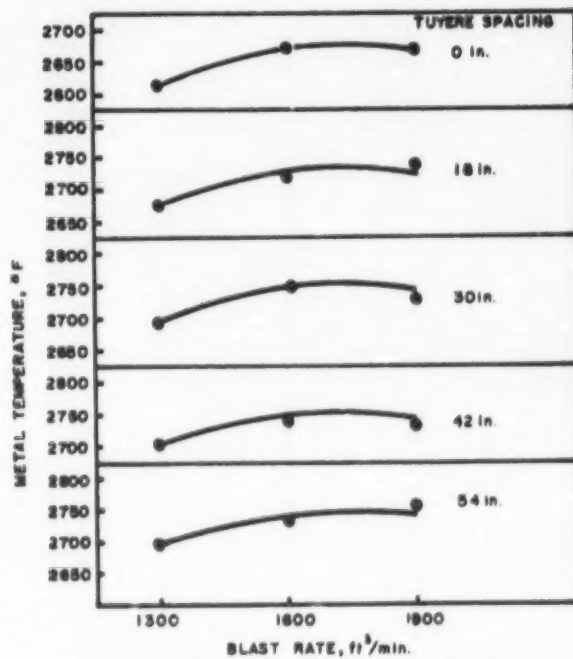


Figure 28. Effect of blast rate on metal temperature at various tuyere spacings. (coke charge 12%) (10)

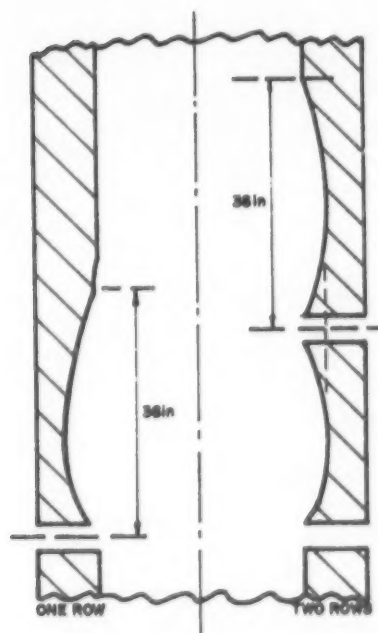
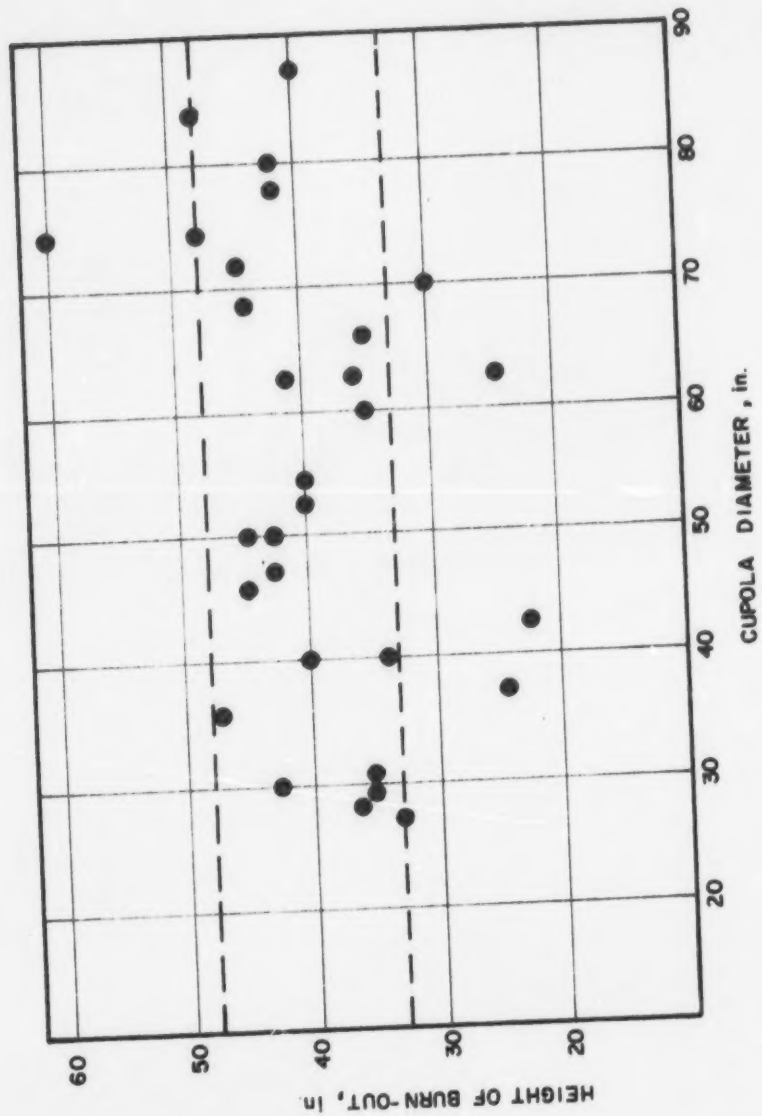


Figure 29. Pattern of lining burn-out when operating with one and two rows of tuyeres. (10)



(10)
Figure 30. Height of burn-out for cupolas of various sizes.

Coke Bed

The use of two rows of tuyeres produces a deeper hot coke bed and so the quantity of bed coke needed is greater than when operating with one row of tuyeres. Due consideration should be given to the extra bed coke required when attempting to arrive at an estimate of the overall saving in coke consumption.

INDUSTRIAL APPLICATION IN THE U.K.

Since the initial tests were performed a number of cupolas have been installed or converted to operate with divided blast and more installations and conversions are planned. Details of divided blast cupolas operating performance reported by eleven foundries and the corresponding information relating to performance obtained previously when operating the cupolas with one row of tuyeres, are given in Table 14.

The cupolas studied vary between 29 and 73 in. (73.7 and 185.4 cm) internal diameter and between about 2 and 25 tons per hour in melting capacity. The distance between the two rows of tuyeres is, in all cases, between 30 and 42 in. (76.2 and 106.7 cm) irrespective of cupola diameter. The proportion of the total quantity of blast air supplied to the lower tuyeres in different installations varies slightly between 50 and 60% and, correspondingly, that supplied to the upper tuyeres varies between 50 and 40%.

Table 14 shows that the benefits of divided blast operations reported by the foundries have, in some cases, been smaller and in others greater than those evaluated under the controlled experimental conditions of operation and measurement at the BCIRA cupola plant. Thus, when advantage has been taken of divided blast operation to reduce charge coke consumption, this amounted to between 2.0 and 5.5% by weight of the metal, in some cases with a reported increase in tapping temperature. On the other hand, where divided blast has been introduced without reducing the charge coke consumption, increased tapping temperatures of 70° to 140°F (40° to 80°C) have been reported. In some instances, other altered features of cupola design or operating practice accompanying the changeover to divided blast operation may have contributed to improvement in performance. Some foundries reported an increased melting loss of silicon with divided blast operation, but at others, this has not been detected.

Some of the plants have been able to take advantage of the increased rate of output made possible by the conversion of existing cupolas to divided blast operation. At others, where coke consumption has been reduced, the cupolas have been operated at a reduced blast rate possibly, in the view of some operations, to the detriment of obtaining the maximum possible saving of coke consumption.

The advantages of using two rows of tuyeres with a divided blast demonstrated at the BCIRA cupola plant have therefore been generally confirmed by the results reported from foundries which are encouraging further adoption of the process as a means of obtaining operating economies.

(10)

Table 14 DIVIDED BLAST OPERATION COMPARED WITH PREVIOUS PRACTICE AT INDUSTRIAL PLANTS

Plant No.	One Row of Tuyeres				Divided Blast				Notes
	Cupola Internal Diameter in.	Charge Coke %	Metal Temperature °F	Production Rate tons/hour	Cupola Internal Diameter in.	Charge Coke %	Metal Temperature °F	Production Rate tons/hour	
1	36	17.2	2820	4.5	36	12.7	2820	5.6	Producing whiteheart malleable iron. Charges consist of pig iron (6-7%), silicon pig (1.7-2.3%), returns 60%, steel scrap 32%. Level of charge silicon increased by 0.2% for divided blast operation.
2	60	14.0	2500	12.3	60	10.5	2500	16.8	Producing iron for ingot molds. Charges consist of 60% heavy ingot mould scrap, 40% pig iron. No change in charge make-up for divided blast operation.
3	44	19.0	2710	7.8	44	13.5	2770	10.6	Producing iron for duplexing for SG iron castings. Charges consist of 70% steel, 30% returns with ferro-silicon briquettes. Level of charge silicon increased by 0.12% for divided blast operation.
4	28	17.5	2370	2.7	33	13.5	2730	4.0	Producing iron for general engineering, machine tool and textile machinery castings. Charges generally contain 30-40% steel scrap.
5	54	12.5	2700	12.9	54	10.5	2700	12.9	Producing iron for automobile castings. Charges consist of 35% pig iron, 40% return scrap, 25% steel. No change in charge make-up for divided blast operation. Silicon content of iron produced 0.1% lower than when operating with 1 row of tuyeres but carbon equivalent value the same.
6	42	14.0	2520	7.8	42	9.0	2640	9.0	Producing iron for ingot moulds. Charges consist of 67% pig iron, 32% ingot mould scrap. No change in charge make-up for divided blast operation. Divided blast cupola operated with closed instead of open slag hole as in previous practice.
7	72	12.5	2460	20.0	72	12.5	2590	25	Producing iron for ingot moulds. Cupola converted to continuous tapping and slagging for operation with divided blast.
8	30	13.4	2560	3.6	30	11.0	2700	3.1	Producing grade 14 iron for general castings. Divided blast applied to new cupolas with increased shaft height of 9 ft compared with previous cupolas. Alterations to charge make-up not connected with divided blast operation prevent comment on effect of divided blast on silicon loss or carbon pickup.
9	73	17.2	2790	16.8	73	13.8	2820	20.1	Producing iron for general engineering castings. Typical charge-25% pig iron, 30% returns, 45% steel scrap. No change in charge make-up when operating with divided blast.
10	12	12.0	2620	1.4	29	12.0	2700	3.9	Producing grade 14 iron for bearing housings and pump bodies. Because of improved casting yield since introduction of divided blast (but not necessarily because of it), charge contains additional 5% steel & 1 additional silicon briquette per 5 cwt charge to replace 5% return scrap.
11	54	18.0	2700	11.8	54	15.5	2700	11.8	Producing castings for automobile industry. Typical charge-32% pig iron, 43% returns, 25% steel. Levels of charge silicon & manganese increased by 0.07%.

NORTH AMERICAN EXPERIENCE WITH DIVIDED BLAST

One of the objectives of this study was to establish a liaison with a number of foundries and individuals actively involved in experimentation and development of divided blast techniques throughout the industry. Among those contacted was Mr. Barry J. Davies, P.E., Manager of Foundry Development for Canron Ltd. of Hamilton, Ontario. The Canron foundry division has been utilizing divided blast on two of their cupolas for approximately two years and in that time Mr. Davies states that they have realized a 27% reduction in their net coke consumption with no appreciable drop in spout temperatures.

Fuel conservation was the primary incentive for instituting the divided blast techniques, however, a number of unexpected benefits accrued as the system was refined. The first, and most obvious, improvement in cupola operation that was noticed was in the visual appearance of the stack emissions. Though no comprehensive studies were done to explain this reaction it would appear that a more efficient combustion of the stack gases, in the upper stages of the cupola, occurred with divided blast. This theory gains merit when it is noted that the stack gas temperature at the exhaust fan was reduced from approximately 550°F to 350°F.(11) This is a very significant development from an energy conservation standpoint for a greater portion of the sensible heat of the stack gas is obviously being utilized in the furnace where it is most useful. This temperature drop is one aspect not highlighted at all in the BCIRA study, and heat losses at the charging doors and from the stack have been an inherent problem with cupolas since their inception. Increased melting rates in the range of 10 to 25% have also been achieved with no appreciable drop in spout temperatures.(11)

Mr. Davies sighted another operational problem which was inadvertently solved with the institution of divided blast. In their operation they were having problems with hot spots near the end of each heat. With divided blast, however, there was a difference in the pattern of refractory burnback. As can be seen in Figure 29 this burnback extended over a higher vertical range, but did not extend as close to the shell as did the conventional cupola burnback pattern, thus eliminating the hot spot problem. This could be a very important factor for foundries where extended heats are necessary to meet production demands.

When the divided blast system was first adopted at Canron the operators experienced a very significant drop in their silicon contents as well as a sizeable reduction in the spout carbon content despite the fact that their temperatures were always maintained at a very high level. After much experimentation Mr. Davies says that they have been able to minimize these losses by maintaining a precisely balanced blast equally divided between the upper and lower tuyeres. He states that, despite arguments calling for 40/60% ratios between the upper and lower tuyeres, it has been their particular experience that even a small amount of extra air through the bottom tuyeres tends to promote increased silicon and carbon losses. Davies further suggests that in order to ensure the equal distribution of the blast

air the divided blast installation should include a new wind belt with an independent row of tuyeres and a separate air weight controller for each wind belt. This would facilitate the maintenance of a constant flow rate to each row of tuyeres by continually correcting for changes in back pressure due to variations in the densities of the individual coke charges along with changes in emission control system conditions during the course of the melt. The Canron facility has employed this type of air control system on their divided blast cupolas and at the present time their silicon yields are approximately 85% and their spout carbons are reported to be very close to the theoretical value.

Another problem which they have encountered during the past two years has been slagging of the bottom row of tuyeres. Mr. Davies has been told that a number of cupolas in the United Kingdom have experienced a similar problem and that by going to smaller sized tuyeres, thereby increasing the velocity of the blast, they were able to alleviate this slagging problem. At Canron they are experimenting with various size inserts in the lower tuyeres and although the problem has not been completely eliminated they are hopeful that continued testing will produce favorable results.

During the last phase of this project arrangements were made for a visit to Canron Ltd.'s Stewart Street plant (the site of the converted cupolas). Mr. Davies and Mr. Bud Meyers conducted us on a tour of the facilities, during which time we familiarized ourselves with the physical layout of the units and discussed the new operating parameters being encountered with the divided blast system.

The two 48" acid lined, cold blast cupolas are located side-by-side, which allows for consolidation of the blast air control system. This arrangement also facilitates the transition from one unit to the other as the cupolas are used on alternate days. Based on the experimental work done by the BCIRA, a 36" spacing was chosen between the upper and lower tuyeres and because of space limitations the new wind boxes are concentrically mounted around the existing wind boxes. There are six tuyeres in each row with the second row on staggered centers above the tuyeres in the first row.

The main blower pipe splits into two separate mains each leading to an individual windbox. The volume of air going through each is controlled with two separate air controllers maintaining a flow rate of approximately 4200 cfm equally divided between the two wind boxes. This instrumentation services both cupolas thus reducing the initial installed cost of the divided blast system, while ensuring a well balanced air supply to both units.

The divided blast system has not created any problems with metal chemistry and carbon pickup and silicon recovery levels have remained consistent with those experienced during the conventional operation. However, Canron is presently producing a very soft iron for pouring large bottom plates (stools) for steel mill ingot molds, and finds it necessary to raise the tapping carbon from 3.20% up to 3.80-4.05%. This is accomplished by blowing graphite with nitrogen through a lance into the iron accumulated in their 10 ton fore hearth.

The addition of a second set of tuyeres requires a corresponding increase in coke bed height. At Canron, this necessitated raising the coke bed by 3 feet, using 3650 lbs. of bed coke instead of the 2600 lbs. used in the original single row operation.

Despite the increase in the size of the initial coke bed Canron has achieved significant savings in raw material costs since they began utilizing the divided blast techniques. The present system has permitted them to drop from 240 lbs. of coke per split to 170 lbs. while maintaining their metal charge at 1800 lbs. This has resulted in a drop from 13.5% by weight of metal, or a metal to coke ratio of 7.4 to 1 down to 9.4% by weight of metal, a metal to coke ratio of 10.6 to 1. In spite of the reduced consumption of coke there was no loss in spout temperature in the 2750 to 2800°F range. No extra splits are required during the 8 to 10 ton per hour melt campaigns which often extend for 14 hours per day. So the net effect has been to reduce the coke charge by 27% even after accounting for the larger initial coke bed.

At the time of our visit Mr. Davies was experimenting with a coke charge of 150 lb. which represents a metal to coke ratio of 12 to 1. The effect of this ratio on metal chemistry has not yet been documented, however, should these results prove favorable, Mr. Davies envisions a total system coke savings on the order of \$250,000 per year.

A look at the overall economic picture shows that the initial capital investment for the conversion of both cupolas to divided blast cost is approximately \$18,000. The installation was easily accomplished during a regularly scheduled shut down. Mr. Davies suggests that \$9,000 per cupola may be a bit higher than usual due to space limitations unique to the Canron Foundry. Since the inception of the divided blast techniques, Canron has achieved an average annual savings of approximately \$170,000 which clearly seems to justify their initial investment.

Additional coke savings are being achieved with the divided blast cupolas by eliminating coke from the last few charges and securing the upper row of tuyeres. By running the cupola with only the single row of tuyeres the cupola operator is able to burn the coke bed almost completely down before dropping bottom.

Mr. Davies firmly believes that, in view of the escalating cost of coke, (which in Canada averages \$100-\$110 per ton) divided blast is certainly a viable approach to fuel conservation. He points out, however, one important factor which must not be overlooked. When you install a new upper row of tuyeres 3 feet above the existing tuyeres, extend the height of your coke bed accordingly; that is, your new coke bed must be 3 feet above the one you are presently using. This increase in coke bed must be amortized over the number of charges that you put into the cupola daily in order to determine your true net coke savings. Therefore, the cupolas having an extended

melting campaign, which is the case at the Canron Foundry, will find the divided blast more economical than those cupolas which use a short melting campaign. It is conceivable that if your melt is short enough, the extra bed coke required would outweigh the savings in coke splits which you could operate successfully with and you could possibly end up with a net increase in consumption of coke.

During the course of the industry survey, it was suggested by a number of foundry managers that York Research Corporation contact Mr. Edgar J. Hopkins of Alabama By-Products Corp., a major coke supplier for much of the southeast. Mr. Hopkins is an acknowledged authority on divided blast systems and has acted as a consultant to several foundries, supplied by Alabama By-Products Corp., during the design, installation and start up phases of their conversion to divided blast.

Mr. Hopkins points out that unlike most cupola modifications the divided blast components can be fabricated and installed by in-house personnel. He further notes that the down-time required, particularly for the smaller cupola applications, can be reduced to a few hours by having all the tuyere and wind box additions pre-fabricated and cut-outs for the existing wind box and cupola shell clearly marked prior to the scheduled shut down.

Mr. Hopkins reports that savings of 25-35% in coke charged, 25-50% savings on refractory due to the change in the burnout pattern, and a 20% increase in melt rate have been achieved by three foundries which he recently assisted in converting to divided blast.(13)

These foundries, Macin Parts Corp. (Anderson, South Carolina), Bahan Textile Machinery Company (Greenville, South Carolina), a subsidiary of MPC, and Cox Foundry and Machine Company (Atlanta, Georgia), make gray and ductile iron jobbing castings in a wide range of sizes. The lengths of their heats vary from two to six hours.

At the time of the conversion, the foundries employed either No. 3½ or No. 4 cupolas, acid lined to a 36" inside diameter. Cold blast was used in all three cases. Two of the cupolas utilized six lower tuyeres. When converted, three of the lower tuyeres were blocked off and raised to a height of 36" above the remaining tuyeres. The third cupola had four tuyeres, and four additional tuyeres were installed 36" above them. The total conversion time for all three cupolas ranged from seven to thirteen manhours. Existing air weight controls and blower systems were not altered.

Each of the cupolas was converted to divided blast using the same formula. The total area for both the upper and lower tuyeres was set at 6.5% of the inside diameter. Of this 6.5%, 40% was allotted to the upper tuyeres and 60% to the lower tuyeres. This uneven distribution was chosen because it is believed that there is less resistance to air flow at the level of the upper tuyeres (due most probably to the difference in density of the charge at that level), making the smaller tuyeres more efficient.(13)

Since the installation of divided blast, the foundries have prepared the cupolas as before with a good bed burn in. However, the total bed height has been raised from 20" to 24" above the old established bed height. This gives a bed height of 48-52".

The charges used by the three foundries in making gray and ductile iron are similar. For gray iron, the charge is 30% steel, 30% pig iron and 40% returns. This provides an in-going analysis of approximately 2.90 to 3.00 carbon and silicon charge of 1.80 to 2.00. Before installing divided blast, the foundries were using a seven or eight to one coke ratio, and now, with divided blast, they are all using a ten to one coke ratio. The result has been improved carbon pickup with the final carbons ranging from 3.50 to 3.70 which is a 60 to 70 point pickup on a 10 to 1 coke ratio. The silicon loss is approximately 20% - about twice the loss encountered when utilizing the conventional single row of tuyeres. With the divided blast, temperatures have ranged from 2780° to 2840°F. (13)

The charge for ductile iron consists of 25% steel, 25% low carbon socal-metal and 50% ductile iron return. Some silicon is charged into the stack as well. This gives a charge going into the cupola of 2.50 carbon and 1.4 silicon. Formerly, a six to one coke ratio had been used to maintain temperature and carbon pickup. Now with divided blast, an eight to one coke ratio or 90 pounds of coke to 750 pounds of metal is used. Temperatures have ranged from 2820° to 2890°F, and final carbons from 3.70 to 3.90. Again a 20% silicon loss has resulted with the final silicons ranging from 1.10 to 1.20. (13)

In addition to coke savings and increased melt rates, the foundries have realized a number of other advantages with divided blast. One benefit which has been noted is that even though the burnout pattern is over a wider area, the total amount of refractory used, during the course of a melt, is less and the foundries are often getting two or three heats before having to reline the cupola.

Another unexpected bonus achieved with the use of divided blast was the realization that even after as much as a 30 minute shutdown, the cupolas recover more quickly than before. With the divided blast, the cupolas are normally back up to 2820°F within five minutes after start up. This reaction appears very similar to that achieved by adding 3% oxygen enrichment to the blast.

An additional plus noted after the installation of the divided blast and when clean scrap is used, is that there is almost no visible plume. Any plume which is visible generally dissipates within ten to twenty feet from the stack. This is attributable, it is believed, to more complete combustion in the upper stages of the cupola as a result of the redistribution of the blast air and the reduction in the coke ash released.

The primary disadvantage reported after conversion to divided blast has been the increased silicon loss which, as noted earlier, is approximately 20% compared to 10% with the single row of tuyeres. It should be noted that, in our earlier discussions with Mr. B.J. Davies of Canron Ltd., he stated that he had experienced similar difficulties following Canron's conversion to divided blast. After much experimentation he found that by installing independent air weight controls for each wind box he was able to reduce the silicon losses to a minimum by insuring a more uniform distribution of the blast air between the upper and lower tuyeres.

While all three foundries had a very good cupola operation before the conversion, the consensus is that the operations have improved as a result of the divided blast. Mr. Hopkins warns, however, that this system is not a cure-all and does not compensate for poor cupola practices.

CONTROL OF METAL COMPOSITION

A major problem generally associated with divided blast systems is a sizeable increase in silicon losses as well as a drop in spout carbon content when compared with the element recovery levels associated with a conventional single tuyere cupola. Much discussion and years of research have been devoted to the solving of these problems, but to date, no clear cut and definitive answers have been formulated. There was, however, an authoritative study presented by the American Foundrymen's Society, in their third edition of "The Cupola and Its Operation." (1) In the discussion, a detailed comparison is made of various stages of combustion (percentages of CO₂ in the effluent gases) and their effect on the composition of the final tap metal.

Silicon and Manganese

The determination of the carbon dioxide content of the stack gases, together with the composition of the metal being tapped at the cupola spout, affords a convenient means of relating the losses of silicon and manganese during the process of melting to the various stages of combustion. A correlation between gas composition and element loss can be obtained from a knowledge of the carbon dioxide concentration of the gases at the time a particular iron sample is taken for analysis.

Based on such data Figure 31 graphically shows the relation between the stage of combustion as determined by the carbon dioxide content of the effluent gases, and the percentage loss of silicon and manganese. The latter figure was obtained by deducting the net silicon and manganese present in the final tap metal from the gross silicon and manganese in the initial cupola charge. These curves show that as the percentage of carbon dioxide (a moderately oxidizing gas) increases, the loss of both silicon and manganese also increases. Conversely, of course, the reverse situation is also true.

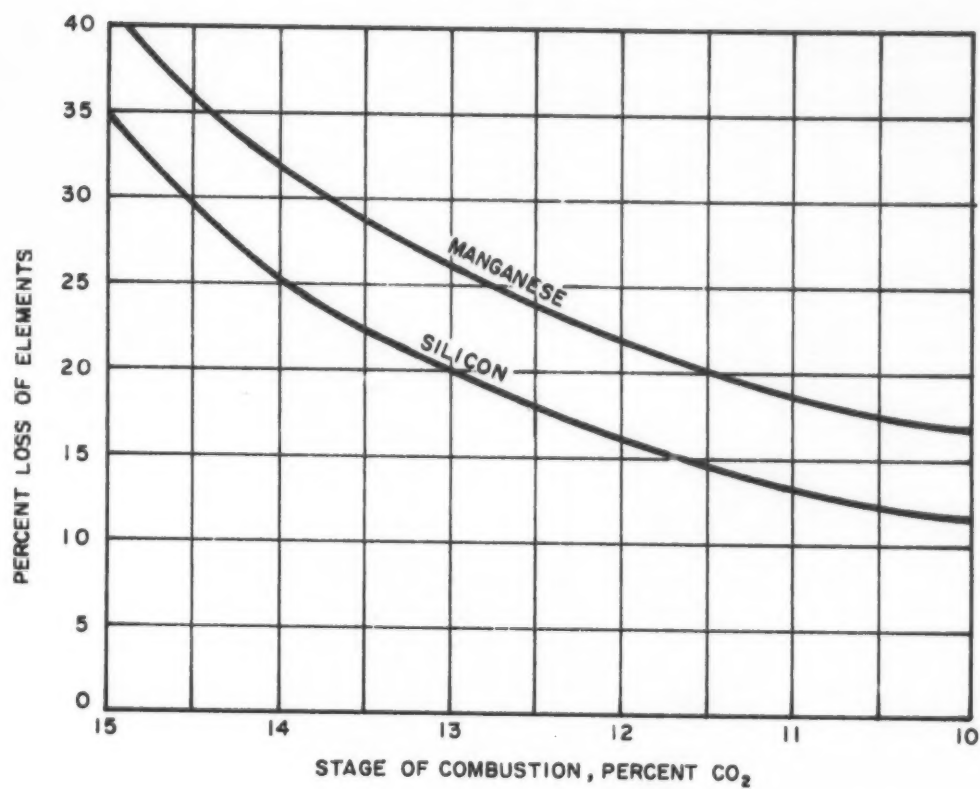


Figure 31. Effect of stage of combustion on element loss in cupola. (1)

From such data it is quite apparent that efficiency of combustion is, in part, restricted by the losses of silicon and manganese when considered from a purely economic viewpoint. It is also obvious that these losses, unless definitely controlled, will have a direct effect on the physical properties of the metal.

Carbon

The actual process by which carbon pick-up in the cupola is accomplished is still a somewhat debateable question. One view is that the process is one of carbonization in which carbon monoxide reacts with the metal to form the carbide. The AFS manual refers to a series of tests performed by Rambush and Taylor which showed that the carbon content of steel bars increased only slightly while they were still solid but increased rapidly as they began to melt. Another study done by Lownie et al was noted which reported similar results and demonstrated that carbon pickup in the cupola well, where the metal is in intimate contact with hot carbon, can account for but a small percentage of the total.

It would appear that the carbon pickup is principally through carburization and is dependent, therefore, on the carbon monoxide content of the gases in contact with the melting and molten metal, hence indirectly a function of the stage of combustion. To add substance to this concept, reference was also made to a series of analyses of metal and effluent gas obtained simultaneously as reported by Massari.

These results, shown in Figure 32, indicate that carbon pickup increases with a decrease in carbon dioxide content (increase in carbon monoxide content) of the gas. Although difficult to accomplish, this data indicates that, within certain limits, the total carbon content of the iron can be controlled if the conditions in the furnace bed can be satisfactorily controlled. Variations in carbon pickup with fuels of different reactivity are probably due to actual differences in the CO/CO_2 ratio in the upper portion of the bed.

ECONOMIC CONSIDERATIONS

When attempting to establish an ideal operating efficiency in a system, such as the divided blast cupola, one must look at the overall effect of manipulating the level of combustion as a method of achieving product control. Just as certain metallurgical benefits result from careful control of the combustion processes of the cupola, so too it can be shown that definite economic advantages can also be realized.

Although not synonymous, melting efficiency (iron to fuel ratio) and combustion efficiency (carbon dioxide content of the stack gas) are closely related. A higher percentage of carbon dioxide in the effluent gases is indicative of the more complete combustion of the coke, and liberation of a greater proportion of the potential heat in the fuel. Consequently, the cost of coke per ton of iron

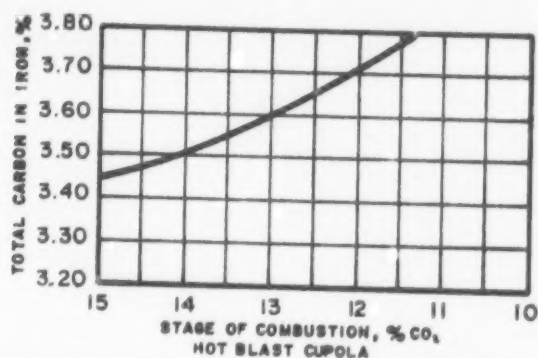


Figure 32. Relationship between total carbon content of the iron and the stage of combustion within the cupola. (1)

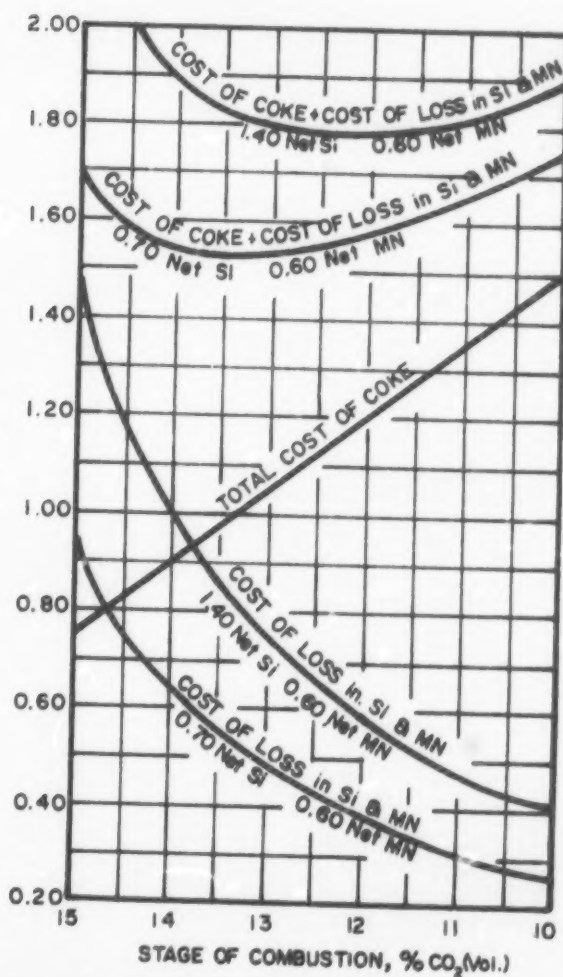


Figure 33. Stage of combustion in the cupola vs. melting. (1)

melted will decrease with an increase of carbon dioxide in the effluent gases, as illustrated in Figure 32. Hence, solely from the standpoint of coke economy, it is evident that a definite savings will result from more complete combustion.

However, it is also shown in Figure 33 that, as the efficiency of combustion increases, the losses of silicon and manganese increase. This means that, as the percentage of carbon dioxide in the stack gases increases, the cost due to silicon and manganese melting losses will increase. Consequently, aside from the metallurgical aspects previously discussed, maximum economy is not attained with maximum efficiency of combustion, since the saving in coke will not counterbalance the cost of the elements lost.

If the summation of the cost of coke and element loss per ton of iron melted is plotted against the stage of combustion (percentage of carbon dioxide in the effluent gases) as in Figure 33, it will be seen that the composite curves pass through an ideal minimum point. These data, for a hot blast cupola, show that maximum economy is obtained when the carbon dioxide concentration of the effluent gases is held between 13 and 14 percent in the case of low silicon iron, and in the vicinity of 12 percent when the metal melted contains 1.40 percent silicon.

When these stages of combustion are achieved for the two irons considered, the most economical quantity of coke is consumed to melt a given tonnage of iron with the minimum cost of alloys. For an equivalent cold blast cupola melting operation, similar economies will result when the carbon dioxide content of the stack gases is from 1.5 to 2.0 percent lower.

In addition to these economies which can be accomplished by combustion control, operation losses are minimized. It is obvious that a control system that ensures greater uniformity of melting will reduce shop losses and rejections.

The capital invested in combustion control equipment can be amortized in a reasonably short time as a result of the various economies which can be affected.

Most authorities agree that it is just as essential to utilize a reliable means of controlling the quantity of air entering the cupola as it is necessary to measure the quantity of coke and iron, and to regulate the composition of the charge. However, the manner in which the air is controlled merits close consideration.

A constant weight of air entering the cupola does not necessarily result in a constant or uniform stage of combustion in the melting zone, since the latter may vary depending upon the type, size, and chemical resistivity of the fuel, in addition to other factors.

A constant volume of air likewise does not at all times contain the same weight of oxygen with which to burn the coke, since the

concentration of oxygen (wt/cuft) will vary with barometric pressure and atmospheric temperature. Similarly, the weight of oxygen delivered will vary with the humidity of the air since any water vapor present replaces an equal volume of air. As a consequence the stage of combustion will, of necessity, vary even if the blast control equipment is designed to maintain a constant volume of air.

In view of the previous discussion relating to the metallurgical aspects and economics of melting, it appears that for optimum operating conditions a more satisfactory control could be obtained by keeping track of the carbon dioxide concentration in the effluent gases and adjusting the blast volume, when necessary, to maintain uniformly the most satisfactory rate of combustion. Though the AFS discussion was geared to cupola operation in general it takes on added significance when related to the element loss problems associated with divided blast systems. There are a number of gas analyzers described in the AFS manual and in view of the vastly improved control technology available today this approach to combustion control may well be the answer to the element recovery problems which have vexed the proponents of divided blast for so long.

SECTION VIII

AIR POLLUTION CONTROL

INTRODUCTION

As stated in the section on cupola operation air pollution control for the cupola is an expensive proposition. It has been shown that hot blast, divided blast and oxygen enrichment can decrease the particulate grain loading from the cupola and can also decrease, by varying amounts, the gas volumes to be treated. Conventional pollution control costs can thereby be reduced significantly. It was decided, however, to briefly investigate innovative pollution control equipment which could be applied to the cupola in expectation of further reducing control costs.

STEAM-HYDRO AIR CLEANING SYSTEM, LONE STAR STEEL

This system is a highly efficient (99%+) pollution control device which can be powered by the waste heat of the cupola. A schematic of the system can be found in Figure 34. The CO laden air is combusted in a waste heat boiler and the steam produced is utilized both to provide the required draft and scrub the particulate. Preliminary particulate removal can be accomplished in the atomizer section. This section has water sprays which can be used to remove the heavier particulate and cool the gas stream. Some agglomeration takes place in this section preparing the smaller diameter particles for the second phase removal. The steam is injected just prior to the mixing tube where collision between the water droplets and the particulate take place. Encapsulation, nucleation and droplet growth occur enabling sub-micron particulate to be brought to a manageable size for removal by low pressure drop cyclone. A shock wave pattern is created in the mixing tube which induces massive turbulent action for maximum scrubbing efficiency.

The device was tested for efficiency by Southern Research Institute under a contract with EPA.(15) To quote from their results:

"The collection efficiency of the Steam-Hydro air cleaning system is quite high. As measured using conventional (Method 5) techniques on a source producing particulate having a mass mean diameter of about 1 micron, the efficiency was measured at 99.90 and 99.84% for two days of testing. Measured fractional efficiencies were about 90% at 0.01 micron, about 70% at 0.05 micron, 85% at 0.1 micron, 99.9% at 0.5 micron, 99.99% at 1 micron and 99.6% at 5 microns. The minimum in the fractional efficiency is probably real, but the actual value is somewhat uncertain because of difficulties in making diffusional measurements in the time variable open hearth process."

Costs

The average capital cost of the Lone Star System is between \$3 and \$5 per SCFM treated depending on the materials of construc-

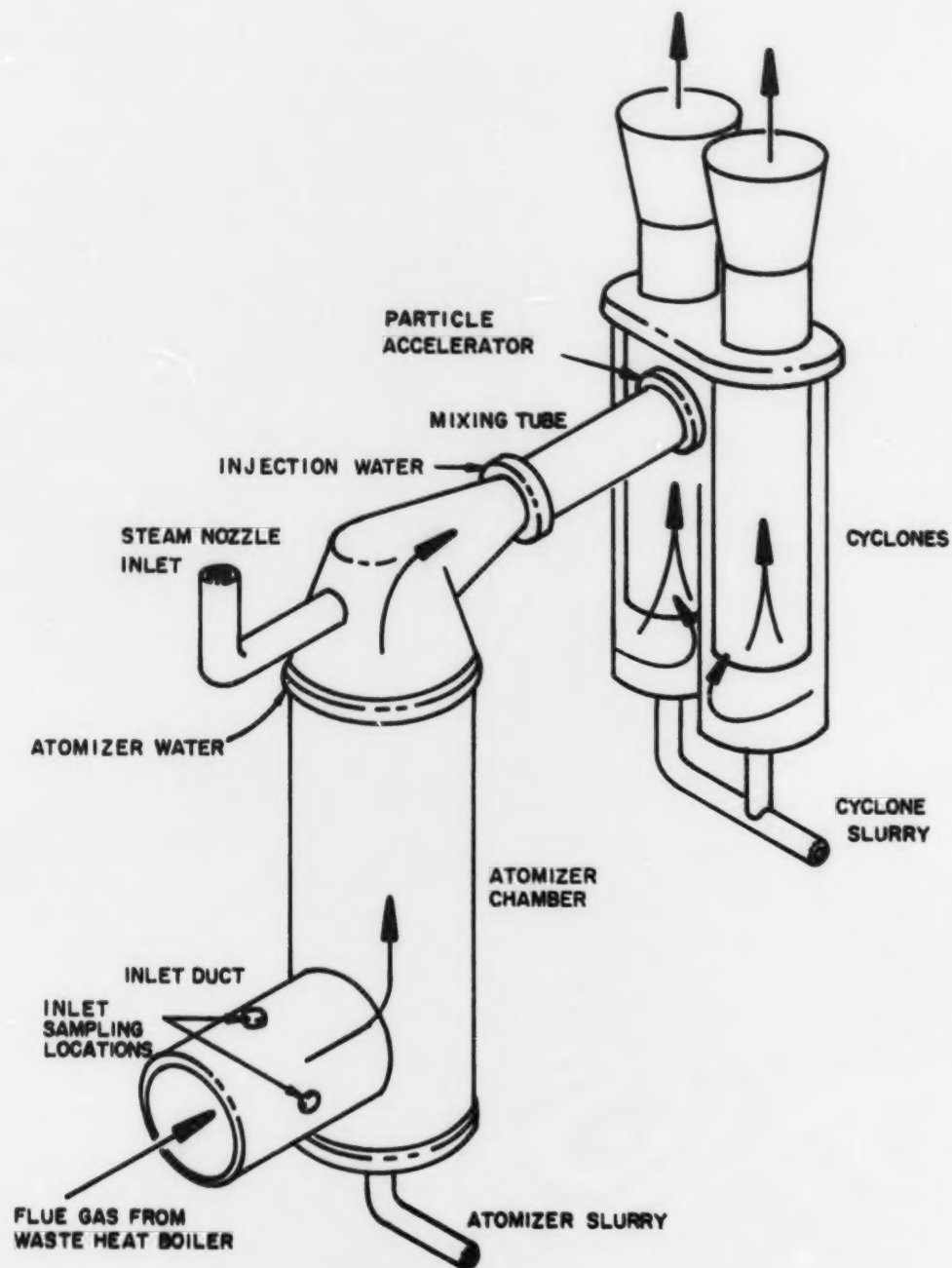


Figure 34. The Lone Star Steel steam-hydro air cleaning system.

tion. Another method of partially estimating capital costs is to determine the size of the waste heat boiler required. Knowing the grain loading required by pollution regulations a graph such as Figure 35 can then be used to determine steam requirements. Since the system pressure drop can also be calculated the steam required to provide the proper draft can be obtained from Figure 36. The higher of the two steam rates is then used for pricing of the waste heat boiler. In the system tested the following condition prevailed:

Steam usage: 7300 lb/hr at 250 psig
Water usage: 61 gal/min (of which 55 gal/min is recoverable)
Air Flow: 13000 SCFM with a system back pressure of 6 inches H₂O
Outlet particulate loading: 0.0007 gr/DSCF

When using the waste heat to power the boiler the only operating cost incurred is the water and pumping cost which would be equivalent to any normal scrubber operation. The primary advantage to this system lies in the fact that an I.D. fan is not required for the draft. For a venturi type scrubber pressure drops of 60 inches of water are common and this necessitates the use of a high h.p. I.D. fan to obtain similar removal efficiencies.

UNITED MCGILL, ELECTROSTATIC PRECIPITATORS

Electrostatic precipitators (ESP) have been rarely applied to the cupola in this country for a number of good reasons. Dust resistivities are high and an appreciable amount of electric power is consumed. The necessity of low velocities for charging purposes increases the size of the unit and the cost. Wire breakages causing shutdowns are common and shutdowns of individual sections of the ESP result in non-compliance with air pollution regulations. The United McGill ESP due to an innovative design overcomes many of these inherent difficulties. The electrostatic field is generated by sharp needles mounted on both edges of the electrode plates. This supposedly allows the ESP to operate at a lower voltage and amperage than conventional units which use suspended wire or rigid frames as discharge electrodes. The discharge electrodes are electrically positive which means that the positive ions used as the charge carriers build up an effective space charge. This would inhibit sparking and arcing to a large degree. As a result the distance between the discharge plate and the collector plates can be decreased enhancing the collection efficiency. If increased collection efficiency is not required, then the size of the ESP can be reduced and the velocity through the unit increased. Since the field outside of the immediate vicinity of the discharge needles is electrostatically stable agglomeration takes place at an increased rate. Another advantage to United McGill's ESP is that there is twice the amount of collection plate area since the conventional wire-plate arrangement is replaced by two plates of opposite polarity. Continual rapping is not required to prevent back corona since the electrostatic plate voltages are not high enough to cause

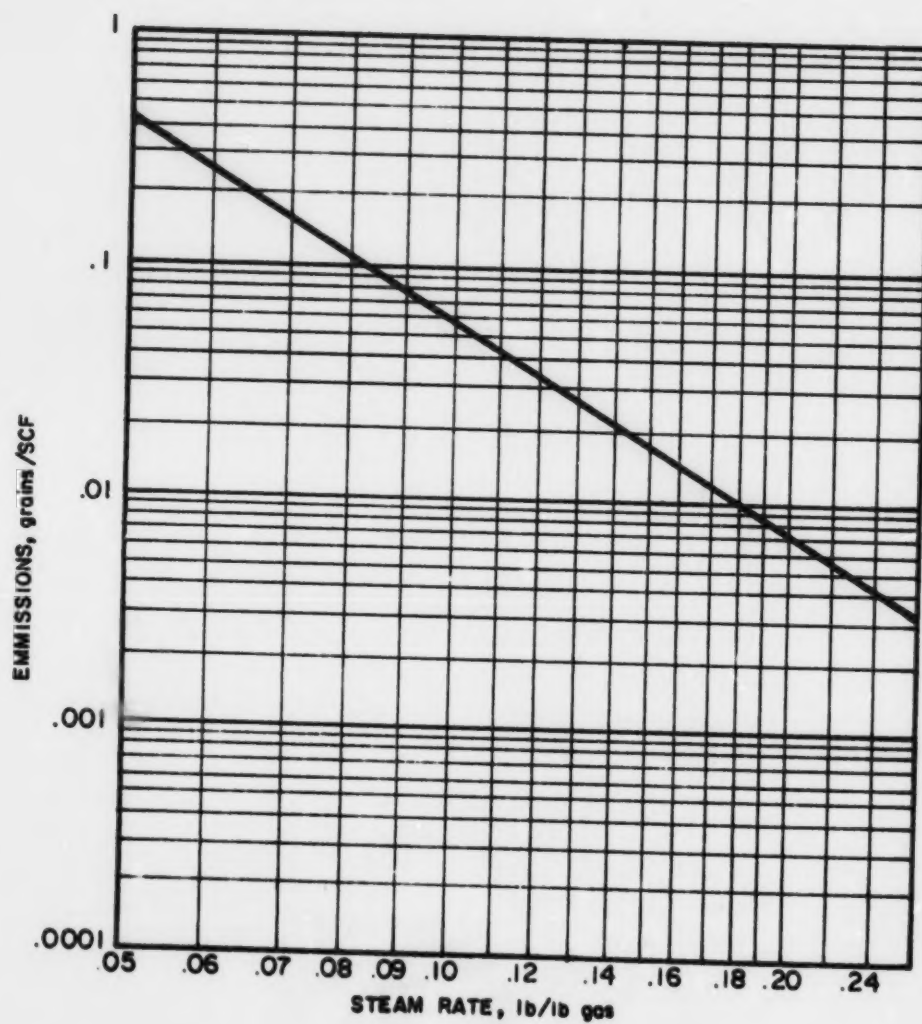


Figure 35. Energy requirement versus emissions

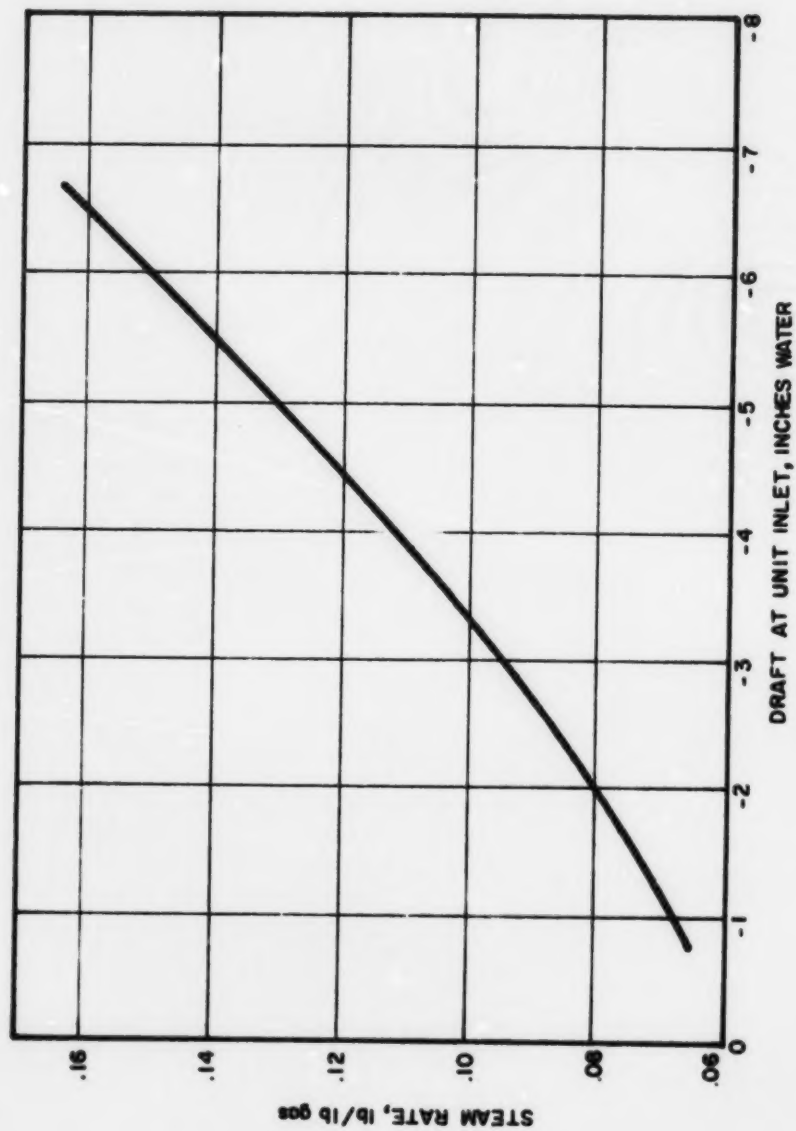


Figure 36. Typical steam rate required for providing draft, Lone Star Steel Company

dielectric breakdown of the dust layer. The needle points, due to their positioning, are self-cleaning and do not need a cleaning system. Since rapping frequency is reduced, reentrainment of the collected dust is consequently reduced. The ESP is usually operated at an inlet temperature of about 500°F and is insulated to protect the materials against acid corrosion caused by cold spots in the system.

United McGill has had success in marketing their ESP in Japan for cupola applications. Due to the innovative design it is believed that the device may well be able to find a market in similar applications in this country.

PACKED BED FILTER SYSTEM

The Canadian Centre for Mineral and Energy Technology has been investigating the use of packed beds for air pollution control. The advantages to such a system are:(16)

1. "Both particulate and gaseous emission abatement can be performed by a single unit."
2. "Hot stack gases may be treated, not only avoiding the problem of gas cooling but also permitting heat recovery from the cleaned gas as well."
3. "If the charge materials and particulate effluent are of a suitable nature, the process can be used to simultaneously preheat charge material and collect and recharge particulate effluent, providing significant savings in both energy and materials."
4. "Considerable flexibility in filtration, pressure drop and gas reactivity characteristics can be achieved by modifications in the sizing and composition of the granular packed-bed materials."

A pilot study was done to determine the feasibility of such a filter arrangement. Figure 37 shows a schematic diagram of the process itself and a cross-sectional diagram of the laboratory-scale unit. The vertical shaft labeled "C" includes inlet and outlet manifolds and inlet and exhaust louvres which control the flow of the bed material in the shaft. A baffle is located within the shaft above the filtration section forcing the bed material to flow against the wall containing the inlet manifold and louvres. The vertical column attached to the top end of the shaft contains the gas entry port "B" and the bed material entry port "A" at the upper end. The hot gases and fines from the bed material are then transported to the bed through ductwork "D". "E" represents a screw-feeder which controls the feed and removal rate of the bed material and collected dust.

Table 15 shows the results of the study and as can be noted the efficiency for the device is quite high. A commercially sized unit (7500 SCFM) is near completion at a foundry in Winnipeg. The system is being marketed under the trade name "Hi-TEC System" by Prasco Ltd. of Canada.

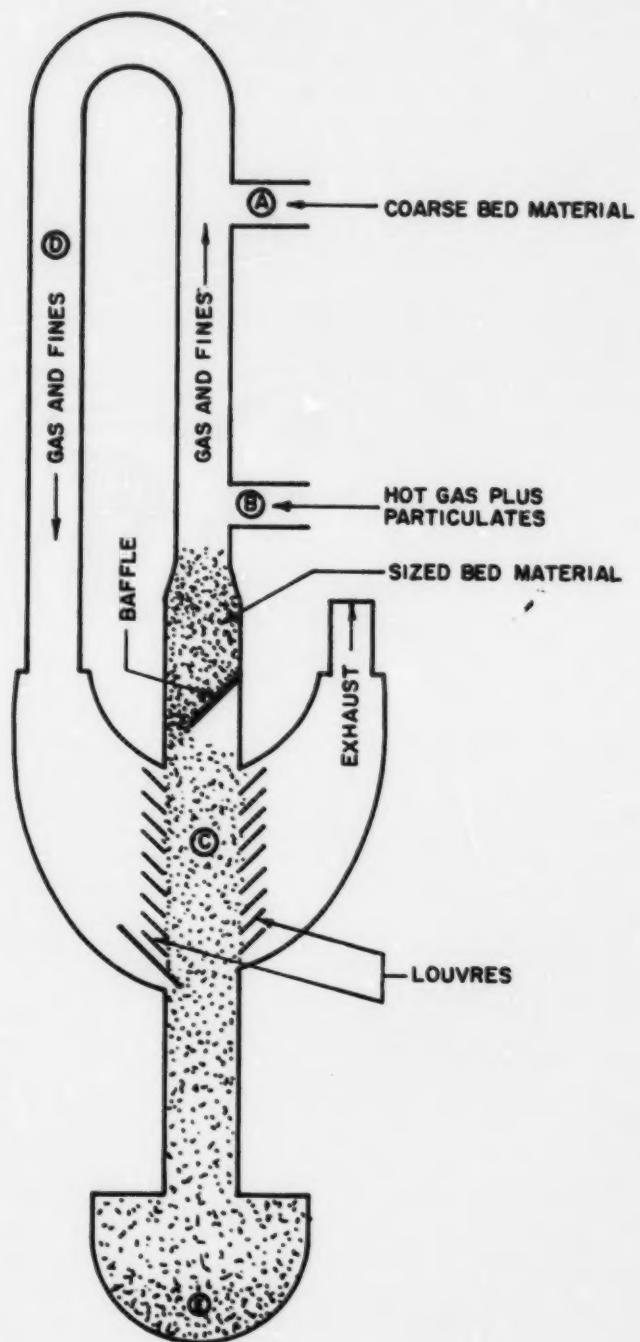


Figure 37. Schematic diagram and cross-sectional diagram of laboratory-scale unit.

Table 15. PRESSURE LOSSES, DUST CONCENTRATIONS, GAS AND BED MATERIAL FLOW RATES, AND FILTRATION EFFICIENCIES FOR A NUMBER OF BED FEED MATERIALS (16)

Packed Bed Filter			Dust Characteristics				
Material		Pressure Drop in w.g.	Type**	Inlet Conc., gr/SCF	Outlet Conc., gr/SCF	Filter Efficiency, %	Gas Flow Rate, SCFM
Type	Rate, lb/min						
M	0.97	2.5-7.5	MCD	135	0.050	99.96	10
M	0.62	5.0	"	134	0.036	99.97	10
M	0.44	7.0	"	134	0.054	99.96	10
M	0.96	5.0	"	45.5	0.175	99.62	15
M	0.98	5.0	"	87.8	0.196	99.78	15
M	0.98	3.5-6.5	"	81.6	0.156	99.81	15
Q	0.62	5.0	MCD	135	0.011	>99.99	10
Q	0.36	5.0	"	132	0.023	99.98	10
Q	0.96	5.0	"	82.4	0.056	99.93	15
Q	1.21	5.0	"	85.1	0.196	99.77	15
Q	0	1.4	GB	56.6	0	100	10
Q	0	1.6	"	65.8	0.14	99.79	10
Q	0	1.9	"	137	1.28	99.07	10
Q	0	2.0	"	150	3.02	97.98	10
Q	0.50	2.2	"	148	4.22	97.15	10
Q	1.22	2.2	"	118	1.19	99.00	10
Q	1.74	2.2	"	153	0.880	99.42	10
Q	2.20	2.2	"	130	0.420	99.67	10
V	1.44	5.0*	MCD	30.5	1.91	93.74	10
V	1.44	5.0*	"	31.4	1.59	94.94	10
V	1.68	5.0*	"	35.7	0.637	98.22	10
BB	0.19	3.5	MCD	141	0.061	99.96	10
BB	0.38	3.5	"	144	0.082	99.94	10
BB	0.76	3.5	"	153	0.109	99.93	10
CC	0.28	3.5	MCD	144	0.010	>99.9	10
CC	0.53	3.5	"	142	0.016	99.99	10
CC	0.89	5.0	"	82.6	0.091	99.89	15
DD	0.04	3.5	MCD	38.0	0.023	99.94	10
DD	0.23	3.0	"	81.2	0.051	99.94	10
DD	0.44	3.0	"	118	0.068	99.94	10

*Pressure varied widely, figure reported is average pressure drop.

**MCD - Mixed Cupola Dust

GB - Glass Beads

SECTION IX

DISCUSSION

This section of the report is intended to give an overview of the technologies investigated and to discuss briefly other techniques to save energy such as waste heat use and general energy conservation.

COMPARISON OF RECUPERATION, DIVIDED BLAST AND OXYGEN ENRICHMENT

To directly reduce coke consumption while maintaining cupola chemistry, a foundry engineer has the three aforementioned technologies to choose from in theory. In fact, however, the choice is usually quite limited by present operation. Recuperative heat exchangers are ideal for future plants and those already with hot blast operation. Divided blast is a cheap, easy method to greatly reduce coke consumption in an existing cold blast operation. Oxygen enrichment has proven itself to be an almost indispensable tool for the foundry engineer but is, in some instances, too expensive to justify on a small scale basis and is frequently misused by operating personnel.

For new installations, recuperators offer numerous different ways to recover the majority of the energy wasted by the cupola. Not only can this energy be used for hot blast operation but a new plant can be designed to use the energy for in-plant heating (as was done at Chevolet Foundry) or for various drying processes. The latter use is possible when the gas is cleaned first and then transported to the combustion site under low oxygen conditions. Oxygen can only be kept at this low level in those systems whose gas take-off point is below 15-20 feet of charge. All of the manufacturers mentioned in the recuperator section market these type systems. In retrofitting a recuperator three problem areas must be considered carefully by the engineer. First, there is a high capital cost involved. Second, ground space should be available for installation. Costs can be sharply increased if superstructuring is necessary. Finally, since about one-half of all cupolas are now externally fired and since these heat exchangers are relatively new, a careful financial analysis must be performed to determine if it is worthwhile to convert to recuperators. In some cases the heat exchangers are designed for natural gas firing only and conversion cost to oil firing (if conversion is possible) should be considered in the costing since cutbacks of natural gas in many areas of the country is now certain.

Divided blast is the best system for those foundries having cold blast cupolas and a need to reduce coke consumption. The techniques recommended in this report can give 20 to 25 percent reduction in coke consumption with a very small capital outlay for an average cupola.

Finally, oxygen enrichment is also capable of reducing coke consumption but in the majority of the foundries which were contacted it was used principally as a combustion tool. The primary disadvantage when compared to the other two techniques is that the operating cost is higher due to the outlay for oxygen. If, however, coke continues to rise in price oxygen enrichment will become increasingly competitive. It is important to note that oxygen enrichment is frequently used as a cure-all for a bad melt by inexperienced operators. Such use can be discouraged by a thorough education of the operators. This caution also holds true for all other techniques. Often the benefits of a new melting technique can be lost if the operator is not properly educated in its use and expected results.

If these three techniques are to be subject to life-cycle cost analysis in conjunction with pollution control equipment, it must be remembered that a decrease in pollution control costs should result from the use of either recuperative hot blast, divided blast or oxygen enrichment. A decrease in air flow results in a decrease in the size of the pollution equipment needed. A decrease in the coke percentages results in a lower grain loading which allows for a slightly less efficient control device to be used.

Waste Heat Use

The energy recuperated for hot blast from a cupola's exhaust gas accounts for less than 50 percent of the energy content of the gas. Among the technically feasible possibilities for use of this energy are the following: in-plant heating, sand and core drying, coke drying, scrap pre-heating, electrical generation, steam and steam generation for compressors and fans. All of these uses are indeed possible. The main drawbacks for actual use are plant layout, low return on investment, and high initial costs. The larger foundries with a number of cupolas and BOF's, while having the initial capital for such schemes, have plant layouts either non-conducive or too limited for actual inception. The smaller foundries on the other hand do not have the capital and could not afford the low return on investment. Probably the only way most of these waste heat concepts will be put into use is in a new plant or by a coincidence in plant layout as in the Chevrolet Foundry.

It is strongly recommended, however, that the Environmental Protection Agency be involved in assisting in the design of any new facility and that all possible efforts be expanded in making maximum possible use of the cupola's waste heat.

Energy Conservation

While attention has been focused on the amount of energy wasted by the cupola, it should be kept in mind that a foundry consumes

and wastes energy in many other ways also. In order to alleviate this the first step to take is to conduct a complete energy audit of the entire facility. Gas, oil, coke, electricity should all be monitored for a set period of time. Often devices such as afterburners are oversized and the gas consumption can be reduced without increasing the CO emissions. Cutbacks on lighting load and rescheduling of shifts to maximize off-peak hours is another way to reduce energy consumption. An energy audit, however, is necessary to systematize the conservation effort. Once the audit is completed those systems which consume the most energy should be carefully checked. If a sizable proportion of the total energy is going to in-plant heating (usually around 7 to 10%) then the plant boiler should be inspected and tuned. Also, conservation methods such as reducing excessive ventilation or leakage should be instituted by the housekeeping personnel. Insulation on boilers and steam lines should be examined and replaced if necessary. Leaks in steam and compressed air lines should be corrected. Old electrical equipment and lighting should be replaced.

While no data is available on how much energy a foundry can save by a conservation program, most engineers agree that for any industry considerable dollar savings can be realized. In many cases foundries do not have personnel trained in energy conservation but contract private consulting firms to conduct a program for them.

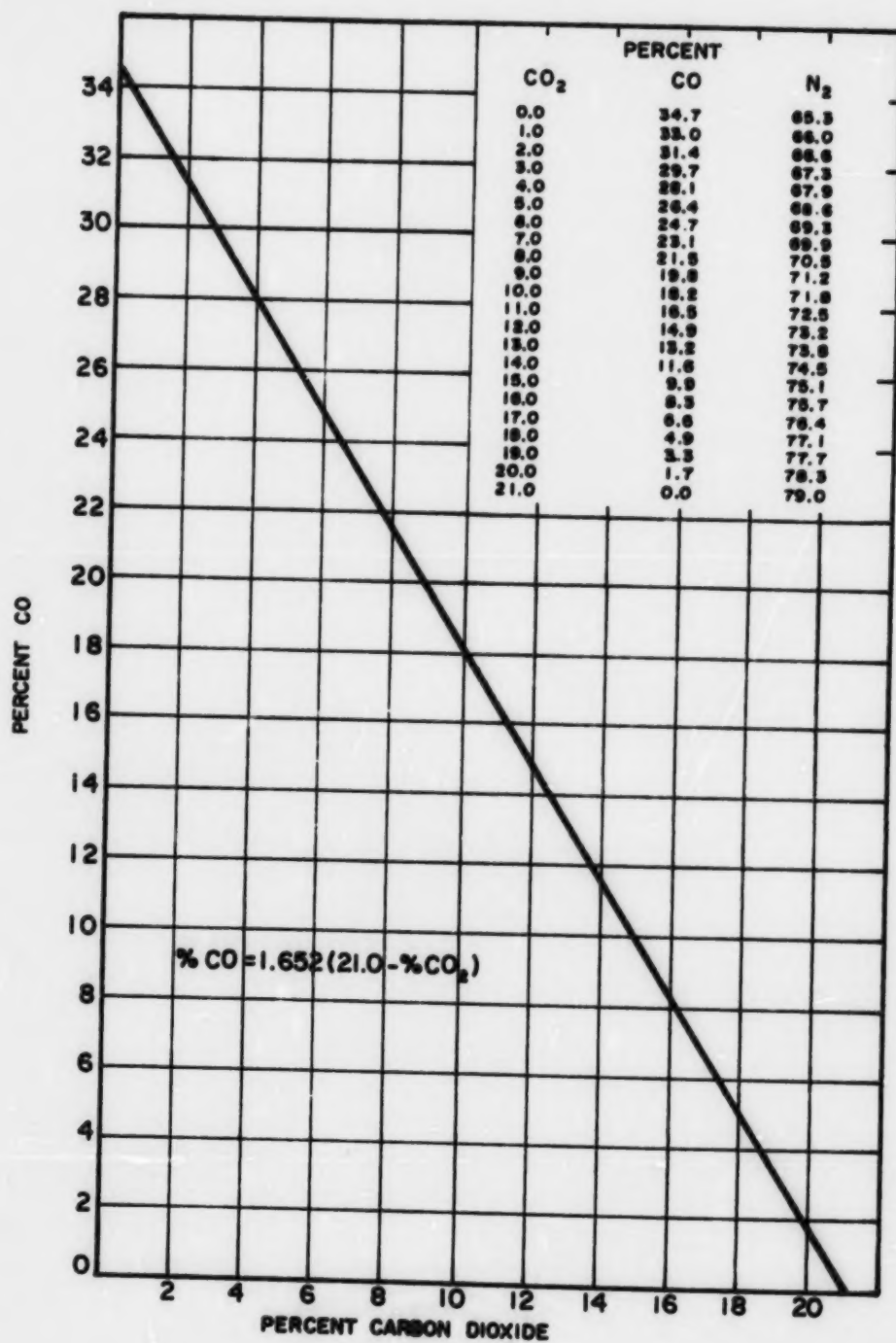
SECTION X

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APPENDIX A



A-1. Relationship between CO₂ and CO in effluent gases.

A-1. MEAN SPECIFIC HEATS

t (°F)	C _p = H/t - 60 Btu per lb per deg F					
	O ₂	H ₂	H ₂ O	N ₂	CO	CO ₂
100	0.2188	3.420	0.4448	0.2482	0.2485	0.2022
200	0.2203	3.434	0.4472	0.2485	0.2488	0.2086
300	0.2221	3.442	0.4499	0.2488	0.2493	0.2145
400	0.2240	3.448	0.4529	0.2493	0.2501	0.2201
500	0.2259	3.452	0.4562	0.2500	0.2511	0.2253
600	0.2279	3.455	0.4597	0.2509	0.2522	0.2301
700	0.2299	3.458	0.4634	0.2520	0.2535	0.2346
800	0.2318	3.462	0.4674	0.2531	0.2549	0.2388
900	0.2337	3.466	0.4715	0.2544	0.2564	0.2428
1000	0.2355	3.470	0.4757	0.2558	0.2580	0.2465
1100	0.2373	3.475	0.4800	0.2572	0.2596	0.2500
1200	0.2390	3.480	0.4844	0.2586	0.2611	0.2533
1300	0.2406	3.487	0.4888	0.2600	0.2627	0.2564
1400	0.2420	3.494	0.4932	0.2614	0.2642	0.2593
1500	0.2434	3.501	0.4976	0.2628	0.2657	0.2620
1600	0.2448	3.510	0.5021	0.2642	0.2672	0.2646
1700	0.2461	3.519	0.5066	0.2656	0.2686	0.2671
1800	0.2473	3.528	0.5111	0.2669	0.2700	0.2694
1900	0.2484	3.538	0.5156	0.2682	0.2713	0.2716
2000	0.2495	3.549	0.5201	0.2695	0.2726	0.2737
2100	0.2506	3.460	0.5245	0.2707	0.2739	0.2757
2200	0.2517	3.572	0.5289	0.2719	0.2751	0.2776
2300	0.2527	3.584	0.5334	0.2732	0.2763	0.2795
2400	0.2536	3.596	0.5375	0.2742	0.2774	0.2813
2500	0.2545	3.608	0.5415	0.2753	0.2784	0.2830
2600	0.2554	3.620	0.5456	0.2764	0.2794	0.2845
2700	0.2562	3.632	0.5496	0.2774	0.2804	0.2860
2800	0.2570	3.644	0.5536	0.2784	0.2814	0.2875
2900	0.2578	3.656	0.5575	0.2793	0.2823	0.2889
3000	0.2585	3.668	0.5614	0.2802	0.2831	0.2902
3100	0.2593	3.680	0.5652	0.2811	0.2840	0.2915
3200	0.2600	3.692	0.5688	0.2819	0.2848	0.2927

A-1. MEAN SPECIFIC HEATS (Continued)

t(°F)	Slag (solid)	Slag (liquid)	Pig Iron (solid)		Pig Iron (liquid)	
			2% Carbon	4% Carbon	2% Carbon	4% Carbon
100	0.1809		0.1200	0.1308		
200	0.1893		0.1223	0.1326		
300	0.1967		0.1245	0.1353		
400	0.2032		0.1275	0.1403		
500	0.2089		0.1292	0.1404		
600	0.2139		0.1309	0.1409		
700	0.2183		0.1327	0.1417		
800	0.2223		0.1346	0.1428		
900	0.2258		0.1367	0.1441		
1000	0.2291		0.1392	0.1457		
1100	0.2321		0.1422	0.1475		
1200	0.2349		0.1458	0.1497		
1300	0.2374		0.1500*	0.1523*		
1400	0.2397		0.1671**	0.1622**		
1500	0.2418		0.1658	0.1616		
1600	0.2437		0.1647	0.1611		
1700	0.2455		0.1638	0.1608		
1800	0.2473		0.1631	0.1606		
1900	0.2490		0.1626	0.1604	0.2208	0.2221
2000	0.2506		0.1622	0.1604	0.2187	0.2197
2100	0.2522		0.1620	0.1604	0.2168	0.2176
2200	0.2538		0.1619	0.1605	0.2151	0.2157
2300	0.2555		0.1618	0.1607	0.2136	0.2140
2400	0.2571	0.3256	0.1618	0.1608	0.2122	0.2124
2500		0.3239	0.1619	0.1611	0.2110	0.2109
2600		0.3224	0.1620	0.1613	0.2099	0.2095
2700		0.3210	0.1622	0.1616	0.2089	0.2083
2800		0.3198	0.1624	0.1619	0.2079	0.2072
2900		0.3187	0.1627	0.1623	0.2070	0.2062
3000		0.3176			0.2062	0.2052
3100		0.3166			0.2054	0.2043
3200		0.3156				

*alpha

**gamma

A-2. HEATS OF FORMATION

Compound	Molecular Weight	k cal per gram mole ^a	Btu per lb mole ^b	Btu per lb ^b
Al ₂ O ₃	101.96	399.09	718,360	7045.5
Al ₂ O ₃ ·SiO ₂	162.05	644.6	1,160,300	7160
CaO	56.08	151.9	273,400	4876
CaCO ₃	100.09	288.45	519,210	5187.4
CaCO ₃ MgO ₃	184.42	556	1,001,000	5430
CaC ₂	64.10	15	27,000	420
CaO·SiO ₂	116.17	378.0	680,400	5857
2CaO·SiO ₂	172.25	538.5	969,300	5627
3CaO·SiO ₂	228.33	688.4	1,239,100	5427
CaO·Al ₂ O ₃	158.04	551 ^c	992,000 ^c	6280 ^c
2CaO·Al ₂ O ₃	214.12	704	1,267,000	5920
3CaO·Al ₂ O ₃	270.20	861	1,550,000	5740
4CaO·Al ₂ O ₃	326.28	1026	1,847,000	5660
12CaO·7Al ₂ O ₃	1386.68	4617	8,311,000	5993
3CaO·Al ₂ O ₃ ·2SiO ₂	390.38	1303	2,345,000	6008
CaO·Al ₂ O ₃ ·6SiO ₂	518.58	1828	3,290,000	6345
4CaO·Al ₂ O ₃ ·Fe ₂ O ₃	485.98	1211	2,180,000	4485
CaS	72.15	115.3	207,500	2877
CO ₂	44.01	94.052	169,294	3846.7
CO	28.01	26.416	47,549	1697.6
Ccoke	12.01	-2.38 ^d	-48,280 ^d	-356 ^d
H ₂ O(g)	18.016	57.798	104,036	5774.7
H ₂ O(l)	18.016	68.317	122,971	6825.6
FeO	71.85	63.7	114,700	1596
Fe ₃ O ₄	231.55	267.0	480,600	2076
Fe ₂ O ₃	159.70	196.5	353,700	2215
FeO·SiO ₂	131.94	276	497,000	3760
2FeO·SiO ₂	203.79	343.7	618,700	3036
Fe ₃ C	179.56	-5.0	-9,000	-50
FeSi	83.91	19.2	34,600	412
FeS	87.92	22.72	40,900	465.2
MgO	40.32	143.84	258,910	6421
MgCO ₃	84.33	266	479,000	5680

A-2. HEATS OF FORMATION (Continued)

Compound	Molecular Weight	k cal per gram mole	Btu per lb mole	Btu per lb
MgO·SiO ₂	100.41	357.9	644,200	6416
2MgO·SiO ₂	140.73	488.2	878,800	6244
MnO	70.93	92.0	165,600	2335
Mn ₃ O ₄	228.79	331.4	596,500	2607
Mn ₂ O ₃	157.86	232.1	417,800	2646
MnO ₂	86.93	124.5	224,100	2578
MnS	87.00	48.2	86,800	997
MnO·SiO ₂	131.02	302.5	544,500	4156
SiO ₂	60.09	205.1	369,200	6144
Na ₂ CO ₃	106.00	270.3	486,500	4590
SO ₂	64.07	70.96	127,730	1993.6
TiO ₂	79.90	218.0	392,400	4911

a. At 25°C.

b. At 60°F.

c. Estimated.

d. Based on P.H. Dewey and D.R. Harper: J. Res., Nat'l Bur. Stds., 21,457 (1938).

A-3. HEATS OF REACTION

Reaction	kcal ^a	Btu ^b	Btu per lb ^c
$C_{\text{coke}} + O_2 = CO_2$	96.43 ^d	173,570 ^d	14,452 /lbC ^d
$C_{\text{coke}} + \frac{1}{2}O_2 = CO$	28.79 ^d	51,820 ^d	4,315 /lbC ^d
$CO + \frac{1}{2}O_2 = CO_2$	67.636	121,745	4,346.5/lbC ^d
$\frac{1}{2}CO_2 + \frac{1}{2}C_{\text{coke}} = CO$	-19.42 ^d	-34,960 ^d	-2,910 /lbC ^d
$H_2O(g) + C_{\text{coke}} =$ $CO + H_2$	-29.00 ^d	-52,210 ^d	-2,898 /lbH ₂ O ^d
$CaCO_3 = CaO + CO_2$	-42.5	-76,500	-764 /lbCaCO ₃
$CaCO_3 \cdot MgCO_3 =$ $CaO + MgO + 2CO_2$	-72	-130,000	-700 /lbCaMg (CO ₃) ₂
$MgCO_3 = MgO + CO_2$	-28	-50,000	-600 /lbMgCO ₃

a. At 25°C.

b. At 60°F.

c. Estimated.

d. Based on P.H. Dewey and D.R. Harper: J. Res., Nat'l. Bur. Stds., 21,457 (1938).

APPENDIX B

CONVERSION FACTORS

Environmental Protection Agency policy is to express all measurements in agency documents in metric units. When implementing this practice will result in undue cost or lack of clarity, conversion factors are provided for the non-metric units used in a report. Generally, this report uses British units of measure. For conversion to the metric system, use the following conversions:

<u>To convert from</u>	<u>To</u>	<u>Multiply By</u>
BTU/lb-F	J/kg-K	4184.
BTU/min	W	17.573
BTU/lb	J/kg	2326
cfm	m ³ /sec	.0004719
OF	°C	5/9 (OF-32)
ft	m	.3048
gal	m ³	.003 x 785
gpm	m ³ /sec	.00006309
hp	W	745.7
in.wc	N/m ²	248.84
lb	kg	0.454
lb/ft ³	kg/m ³	16.018
oz/in ²	N/m ²	430.922
psig	N/m ²	6,894.757